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Premining Investigations for Hardrock Mines

Proceedings: Bureau of Mines Technology Transfer Seminar, Denver, Colo., Sept. 25, 1981

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PREFACE

This Information Circular summarizes the results of recent Bureau of Mines research concerning improved methods and techniques employed during premining research for metal and nonmetal mining. The papers are only a sample of the Bureau's total effort in the areas of ground control, maximum resource recovery, and efficient extraction technology, but they delineate the major advances in the area of premining research. Much of the technology discussed has been refined from previously used techniques and is applicable to other areas of mining and to other industries such as petroleum.

The technical presentations reproduced herein were made by Bureau technical personnel of the Technology Transfer Seminar on Premining Investigations for Hardrock Mines given September 25, 1981, in Denver, Colo. Those desiring more information on the Bureau's research in the areas described or other general information should contact the Bureau of Mines Technology Transfer Group, 2401 E Street, N.W., Washington, D.C. 20241, or the appropriate author.

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PREMINING INVESTIGATIONS FOR HARDROCK MINES

Proceedings: Bureau of Mines Technology Transfer Seminar,
Denver, Colo., September 25, 1981

Compiled by Staff, Bureau of Mines

ABSTRACT

These proceedings consist of papers presented at a Bureau of Mines Technology Transfer Seminar in September 1981 for the purpose of disseminating recent advances in mining technology in the area of premining research. The introduction and descriptive papers discuss techniques and instrumentation used in premining research for metal and nonmetal mining and shaft design and borehole control for premine planning.

INTRODUCTION

By Joseph L. Condon[1]

The Bureau of Mines traditionally defines premining investigations as the series of studies that follow an exploration discovery and precede production from a mineral deposit. In the mining industry the studies may be the responsibility of a specialized group within a company, or they may be shared between exploration and production organizations. Typically, the studies at one prospect are sequential, with the start of a successive study dependent on the successful conclusion of a prior study, until both economic and engineering feasibility of production from the deposit are demonstrated. Beyond that point in time, more information is necessary to describe the characteristics of the ore body for development and detailed mine planning. After mining has started many techniques developed for premining investigations are usable to define the ore body beyond the working face for production planning.

The Bureau of Mines research in premining investigations is conducted under the Assistant Director--Mining Research. It includes the development of methodologies for collecting and analyzing information on ore body characteristics to enhance mine production and safety. The aim of the Bureau research program is not the development of technology for its own sake, but to develop innovative methods that are better and cheaper than existing techniques to replace and supplement current industry practice and to improve conventional techniques to increase their capabilities and lower their costs.

The Technology Transfer Seminar on Premining Investigations for Hardrock Mines describes and demonstrates the results of recent Bureau research. Pulsed and continuous wave ground radar systems that are capable of mapping ore body characteristics with high resolution have been shown feasible for application from the surface, in boreholes, and from the working face underground. High-resolution seismic methods are now available that are capable of utilizing higher frequencies and obtaining geologic information from much shallower depths compared with the conventional seismic reflection technology for the petroleum industry. Experience with coal strata has shown that the high-resolution seismic reflection method can map subsurface features in media unfavorable for the propagation of electromagnetic radiation, while the ground radar method can provide superior details within the coalbed. Analogous application of the two techniques is anticipated in hard-rock mining.

A computer program has been developed to map lineaments such as geologic faults and fractures from satellite imagery. Past research has demonstrated that ground control problems are often present in underground mines below the intersection of lineaments that are apparent in remote sensing data. Readily available and relatively low-cost resistivity equipment is usable for fast reconnaissance to verify remote sensing data.

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[1] Research supervisor, Denver Research Center, Bureau of Mines, Denver, Colo.

An in situ neutron activation analysis system has been developed to assay ore in place from small-diameter access boreholes. The logging sonde in the system has a 4,000-channel, microprocessor-controlled analyzer with digital communication over a standard logging cable to the surface. The system permits a laboratory-quality gamma ray spectrometry from several thousand feet deep in the earth.

A viable diamond core bit system that can be replaced inside the drill hole has been developed to significantly improve the efficiency of core drilling by minimizing the time and labor required for bit replacement. Techniques for controlling the deviation of drill holes have also been developed. Even with the availability of in situ assaying technology, core drilling will be required for "hands on" samples and for that reason alone will never be totally replaced.

Finally, a method to design deep shafts in hard rock will be discussed. The design technique relies partly on geotechnical data that can be obtained with existing instruments and mechanisms.

The results of Bureau research in the subject area are impressive. With the new techniques available, the mining industry should improve its practices for premining investigations. The Bureau of Mines continues to provide real solutions to meet the challenge of varying geologic conditions and changing mining technologies for various types of mineral deposits.

HIGH-RESOLUTION SEISMIC METHODS FOR HARD-ROCK MINING

By Frank Ruskey[1]

ABSTRACT

Seismic procedures for applications in mining problems are continually improving. Increasing numbers of mining companies are using the technology or exploring its possibilities. This paper, primarily intended for the casual user, describes techniques and procedures, constraints, and pitfalls that may be encountered. Results of tests at various sites are presented to show the possibilities of the technology.

INTRODUCTION

This report provides the mining industry with the results of research and evaluation performed by the Bureau of Mines in high-resolution seismics for mining applications. The work arose out of a need to find and develop a geophysical technique that could locate channel sands and faults from the surface for mine safety planning. From an evaluation of various geophysical methods, magnetics, electrical, gravity, and seismic, the seismic method was soon considered the best tool for locating geologic features related to mineral deposits, such as faults and channel sands. The other geophysical tools have their unique value for other mining applications.

Once the value of seismics was established, it became apparent to Bureau of Mines researchers that a multifold approach to the problem was necessary. In-house research was considered essential to develop the concepts towards achievable goals that could be used by the mining industry. Coupled with this was a need to stimulate potential users into realizing the value of seismics for improving safety and providing extraction economies. It was believed that the mining industry would be interested in the viability of using seismics for solving mining problems, and that geophysical service companies could be made aware that a potential market exists for their services. Accordingly, contracts were let over several years to test feasibility, configure an optimum geophysical data acquisition system from off-shelf equipment, and perform tests in many mining environments and for a variety of mining problems. Commercial seismic processing centers were contracted to analyze data, using their already developed oil exploration expertise. They were chartered to test a variety of techniques, to determine which were best for the many possible types of mining problems, and to simplify each technique so that a practical, economical service can be engendered. In addition, Bureau personnel have presented the results of their work at many conferences and symposiums to let the mining industry know that, indeed, here is a tool that can save both lives and, potentially, millions of dollars in production costs.

Also, it has been recognized that there are many problems associated with mining other than fault and channel sand delineation that can be aided with applications of seismic technology. Among these are abandoned mine location, ore zone boundaries, split coal seams, vein delineation, and others. Some of these have been the subject of preliminary investigations by Bureau researchers. Some of these results are described herein to indicate the full potential of the seismic technique. Although the field examples presented in this report were taken from tests in coal, the procedures are identical with those used for any ore deposit in any geological environment.

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[1] Geophysicist, Denver Research Center, Bureau of Mines, Denver, Colo.

The report includes, in addition to Bureau work, that of some Bureau contractors over the past 5 years. Their pioneering efforts have been significant in bringing seismics for mining applications to its present usefulness. Additionally, Sheriff (13)[2] presented a brief description of commonly used seismic terms.

RECOMMENDED SPECIFICATION FOR DATA COLLECTION AND RECORDING SYSTEM

This specification is based upon the field experience of Bureau of Mines personnel over the past 5 years. It is intended as a guide to potential users or purchasers in their shopping, or as a basis for evaluating proposing seismic service companies that may be under consideration to perform work for their mining company. While it is not intended to lock out potential service companies that would be able to solve many problems and achieve useful results using less sophisticated equipment, the recommendation is that a higher ratio of success can be anticipated, particularly for knotty problems or terrain conditions, if the following minimal criteria are met.

Data Acquisition--Minimal Specifications

1. Digital System: A digital system is a must. Data sample rates of 1 mil are acceptable for most applications. Sample rates of 0.5 mil or 0.25 mil will aid higher resolution. This is shown cursorily in table 1.

TABLE 1. - Table of recordable frequencies at various sample rates

Sample rate, mil	Frequency, Hz	Approximate resolution, ft	Approximate depth of applicability, ft
2	128	4	1,000-10,000
1	256	2	300- 3,000
.5	512	1	100- 500
.25	1,024	1	200- 300
.125	2,048	.5	50- 200

2. Stacking Capability: For most applications, the data acquisition system should be able to stack up to 20 repeat shots, if necessary. An acceptable system, but cumbersome, would be one where each shot or stack element is recorded on tape and stacked at the processing center.

3. Visual Recording for Field Monitoring: The field operator should be able to look at his field data and make judgments on whether to alter his field procedure, for optimal signal returns, as subsurface conditions change. Simultaneous visual recording of all traces is desirable. Trace by trace recording is acceptable, but slow and cumbersome.

4. Amplifier Gain: Each recording channel should have 96-db gain, in approximately 12-db steps, to enable sufficient gain over a full spread, and to taper the spread gains from near-shot to far geophone positions.

5. Record Time: The system should record a minimum of 0.5 sec at the 0.5-mil sample rate.

6. Gain Ranging: This feature (sometimes called automatic gain control) can be offset for shallow work, if sufficient initial energy can be put into the ground. If low-level inputs are used, or much stacking is required, then gain ranging is desirable.

7. Magnetic Tape: The data must be digitally recorded for later computer processing. The subtleties of shallow seismic analysis can only be brought out

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[2] Underlined numbers in parentheses refer to items in the lists of references at the end of each paper.

with computer processing. Computer-compatible reels are most convenient for mass storage volume and for later ease of mounting on data processing equipment.

8. Seismic Detectors: Geophones (velocity detectors) or seismic accelerometers are equally good. Geophones have proven designs, ruggedability, reasonable response, low cost, and minimal maintenance. Accelerometers require line amplifiers that need periodic maintenance or battery changing. The use of subarrays will compromise resolution to a degree, but they provide greater field versatility for improved signal-to-noise capability over single geophone layouts.

Alternate Systems

Although this report makes a strong case for sophisticated data acquisition systems, it should be kept in mind that many of the simpler systems now on the market can yield significant data for problem solving. The use of such systems is recommended where budget, terrain, or personnel constraints inhibit the use or acquisition of the more expensive gear.

Additionally, systems are being developed that provide almost immediate processing of the filed data. Such systems are cost effective and will provide a useful tool to the coal company that wishes to perform its own surveys at minimal cost. One such economic system is noteworthy of description.

As part of their Coal Geophysics Research Project, the U.S. Geological Survey is developing a small-crew, relatively inexpensive shallow target seismic system. Figure 1 shows the four prototye elements of the recording system; a 12-channel, signal-enhancement, 10-bit seismograph, with sample rates from 0.05 to 2.0 msec; an interface box containing filters and signal-switching units; a 12-channel monitoring oscillograph; and a digital tape recorder. A special one-wheel carrier is used to first move a posthole-type drill to the site, and then bring in the recording equipment. With its handles turned down, this carrier becomes the covered table upon which all elements of the data acquisition system are mounted.

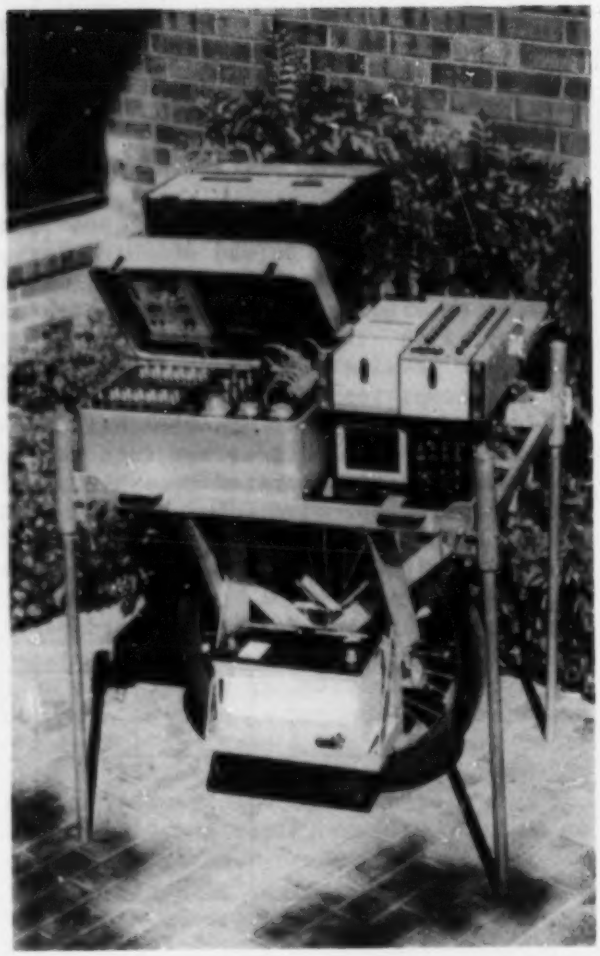


FIGURE 1. - Inexpensive, all-purpose shallow seismic system. *(Courtesy, U.S. Geological Survey.)*

Data are processed using the desk-top computer system shown in figure 2. This equipment is carried in a small trailer. By having the computer system located near the center of field operations, it is possible, on the evening of the day on which the data were taken, to review their quality and to make an initial evaluation of results.

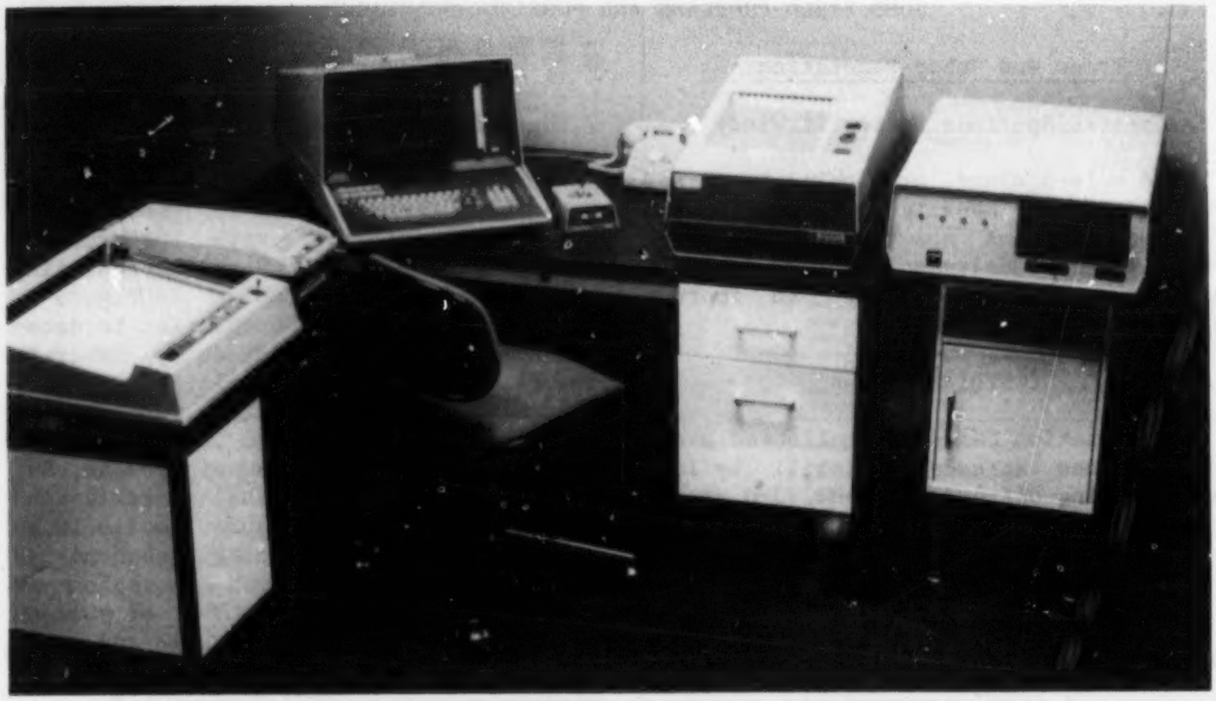


FIGURE 2. - Data Processing system for U.S. Geological Survey seismic system.
(Courtesy, U.S. Geological Survey.)

FIELD TECHNIQUES

This section provides a partial description of field techniques used by Bureau personnel with reasonable success. It also includes some information on procedures used by others, primarily commercial crews, that are known to be successful. The coverage of all procedures is intended as a user's guide only and recognizes that there may be many other successful techniques under wraps, or as yet unproven. Greatest success will be achieved by operators that have the versatility to use several procedures, in order to tackle problems where geologic conditions vary. Optimum field techniques are field condition dependent. Geologic conditions, terrain conditions, groundwater, and cultural conditions may each have to be approached with a different procedure.

Geologic conditions include the lithology of the deposit and the overlying strata and structural conditions, such as faulting, facies changes, or unconformities. Each of these will affect seismic signal returns, and one should be at least generally aware of their likelihood in the survey area.

Terrain conditions include trees and other vegetation; hills, valleys, streams, or other bodies of water; swampy versus dry land; and heavy, spongy soil mantle versus a thin mantle or exposed outcrop rock. Included here also are access roads, habitation, buildings, roads, and pipelines, etc. Each of these has its adverse effect on a proposed survey. However, each condition can be reasonably tackled with a versatile crew, provided they are allowed the time to tailor their approach to changing conditions.

The following section is presented to indicate some of these problems and some potential solutions.

SOME FIELD PROBLEMS AND POSSIBLE SOLUTIONS

Trees and Other Vegetation

Creates Spurious Noises If Windy

Wind noises alone, or rocking trees and vegetation, are usually low frequency and random; solutions are to use (a) higher frequency (100 Hz) geophones and (b) repeat shots with stacking to improve signal-to-noise response.

Inhibits Access

If vegetation cannot be bulldozed away, the geophone spreads can still be laid out by using off-end, fan, or 3-D shooting techniques.

Hills and Valleys

Source of Spurious Reflections That Mask Wanted Seismic Signal Returns

The spurious reflections usually show up as a strong band in the seismic section. Computer processing migration techniques are effective in subduing this source of noise. In extremely hilly terrain 3-D or concentric circle techniques described in later sections of the report may be used effectively.

Changes in Lithology

The effect of changes in lithology is shown hypothetically in figure 3, where it may be seen that the longer travel path from shot-point B to geophone B' reduces compatibility with the results from shot-point A. Hence, a confusing seismic section, as shown in figure 4, may ensue. Part of the solution is data-processing-center dependent, wherein traces with abrupt elevation changes, used for common depth point (CDP) stacking, are trace analyzed and adjusted. This may be a tedious manual process, but it is necessary for shallow reflection work. In the field, some amelioration can be achieved by placing geophones at nearly the same contour elevation--if possible, or by using the 3-D or concentric circle techniques. Static and moveout corrections have to be applied with discretion.

Inhibits Access

If the seismic crew is vehicle dependent with drills, recording truck, shooting or energy source trucks, they will be limited to available access roads. For such cases 3-D or offset shooting can be

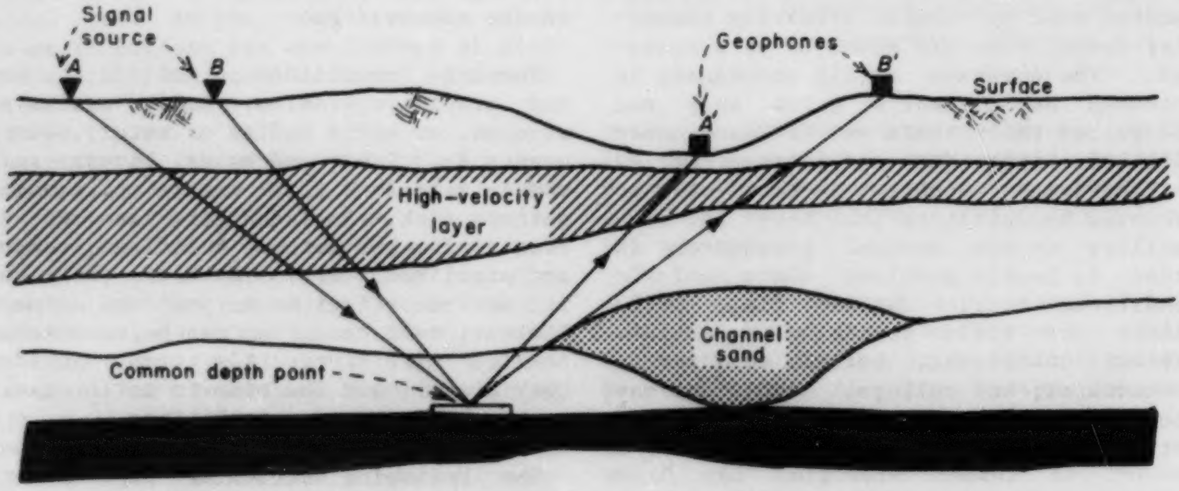
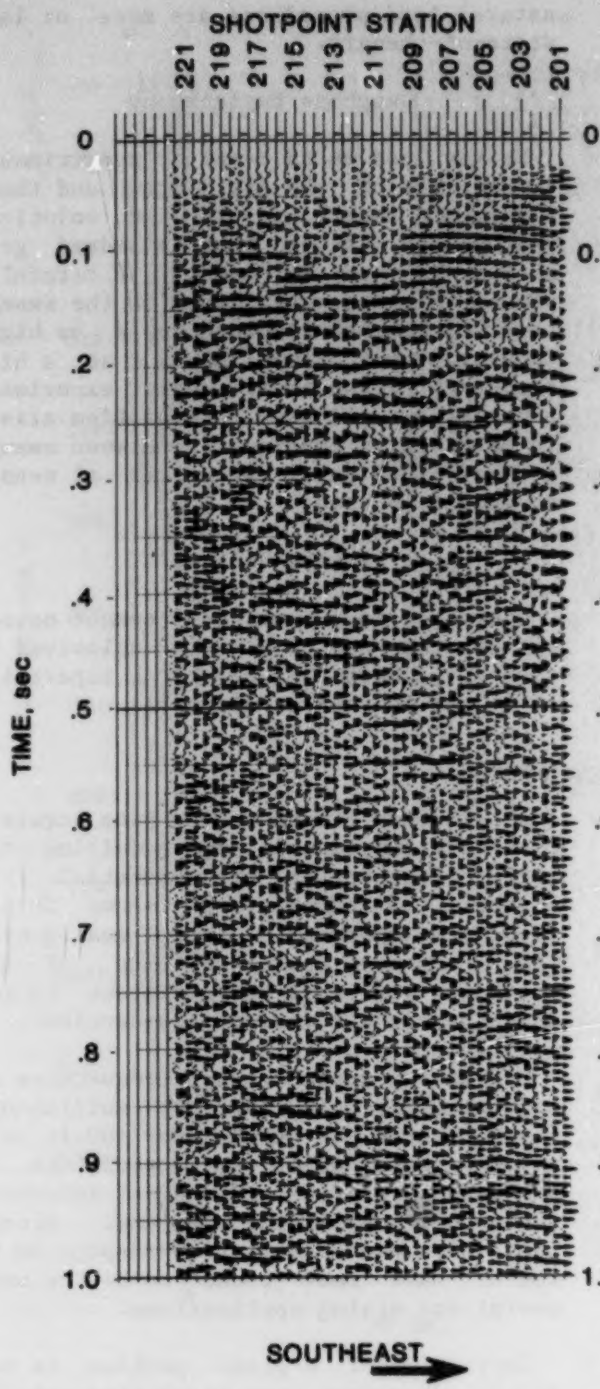
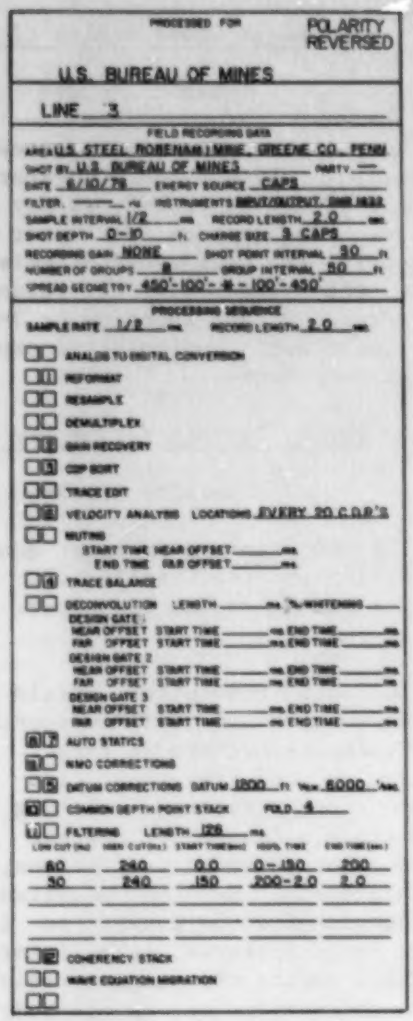


FIGURE 3. - Lithologic changes that may complicate seismic data returns.





PROCESSED FOR

POLARITY REVERSED

U.S. BUREAU OF MINES

LINE 3

FIELD RECORDING DATA

AREA U.S. STEEL ROBENA No. 1 MINE, GREENE CO., PENN
SHOT BY U.S. BUREAU OF MINES PARTY —
DATE 6/10/78 ENERGY SOURCE CAPS
FILTER — Hz INSTRUMENTS [illegible]
SAMPLE INTERVAL 1/2 ms RECORD LENGTH 2.0 sec
SHOT DEPTH 0-10 ft CHARGE SIZE 3 CAPS
RECORDING GAIN NONE SHOT POINT INTERVAL 50 ft
NUMBER OF GROUPS 8 GROUP INTERVAL 50 ft
SPREAD GEOMETRY 450'-100'-*-100'-450'

PROCESSING SEQUENCE

SAMPLE RATE 1/2 ms RECORD LENGTH 2.0 sec

☐ ☐ ANALOG TO DIGITAL CONVERSION
☐ 1 REFORMAT
☐ ☐ RESAMPLE
☐ ☐ DEMULTIPLEX
☐ ☐ GAIN RECOVERY
☐ 3 CDP SORT
☐ ☐ TRACE EDIT
☐ ☐ VELOCITY ANALYSIS LOCATIONS EVERY 20 C.D.P.'S
☐ ☐ MUTING
START TIME NEAR OFFSET ____ ms
END TIME FAR OFFSET ____ ms
☐ ☐ TRACE BALANCE
☐ ☐ DECONVOLUTION LENGTH ____ ms %-WHITENING ____
DESIGN GATE 1
NEAR OFFSET START TIME ____ ms END TIME ____ ms
FAR OFFSET START TIME ____ ms END TIME ____ ms
DESIGN GATE 2
NEAR OFFSET START TIME ____ ms END TIME ____ ms
FAR OFFSET START TIME ____ ms END TIME ____ ms
DESIGN GATE 3
NEAR OFFSET START TIME ____ ms END TIME ____ ms
FAR OFFSET START TIME ____ ms END TIME ____ ms
☐ ☐ AUTO STATICS
☐ ☐ NMO CORRECTIONS
☐ 5 DATUM CORRECTIONS DATUM 1800 ft Vel 6000 ft/sec
☐ ☐ COMMON DEPTH POINT STACK FOLD 4
☐ ☐ FILTERING LENGTH 128 ms

LOW CUT (Hz)	HIGH CUT (Hz)	START TIME (sec)	TAPER TIME	END TIME (sec)
60	240	0.0	0-150	200
30	240	150	200-2.0	2.0

☐ ☐ COHERENCY STACK
☐ ☐ WAVE EQUATION MIGRATION
☐ ☐

400% COHERENCY STACK, polarity reversed

FIGURE 4. - Marginal-quality seismic section due to underdeveloped field procedures.

considered. A fully portable system, such as developed by the USGS Coal Branch, *previously* described, would be useful for these conditions.

Streams or Other Bodies of Water

Inhibits Access

For most applications access is the problem. If the body of water is large, shallow marine procedures may be necessary. The stream or water body may restrict the laying out of seismic lines, or the placement of signal energy sources. Offset shooting or 3-D shooting are both well developed and provide practical approaches.

Swampy Terrain Versus Dry Land

Inhibits Access

The solution to swampy conditions is similiar to that applied to streams or other small bodies of water. The dry natural land procedures are more or less state-of-the-art.

Geophone Emplacement

Swamps can be an asset or a detriment. Versatility of the field crew and their equipment repertoire is the solution. Hydrophones, rather than standard geophones, can now be used. If carefully placed into the water level of the swamp, their frequency response could be high. However, some swamps may act as a high frequency attenuator. Only experience and testing will tell. A problem arises in maintaining continuity between swampy and dry land data, if a mix of sensor equipment is used.

Signal Sources

Operating in a swamp environment necessitates the use of either explosives or sparkers. Here the need for a repertoire of signal sources is essential.

USEFUL FREQUENCIES

In high-resolution field surveys, the question of optimum source-receiver frequencies becomes an issue. High frequencies and high resolution are interdependent. Factors affecting choice of recordable frequency content are cost, speed of seismic productions, depth of penetration, and the magnitude of the geologic feature sought in the survey. For large features, such as large channel sands, faults with considerable throw, or the boundaries of an ore body, a lower frequency survey will be adequate. For small features, such as ore veins, splitting coal seams, small channel sands, or faults, a higher frequency survey is necessary.

Equipment choices for high-frequency surveys become a factor. High- and low-frequency energy can be obtained, for instance, from explosives, controlled explosion chambers, and weight drops. Detector responses have to be compatible with the anticipated frequency spectrum. Standard geophones, with a natural frequency above 40 Hz, are recommended to help attenuate the inevitable low-frequency ground roll. Data acquisition systems capable of acquiring the higher frequencies are essential. If digital, the sample rates shown in table 1 are important. If an analog system, response characteristics of the recording pen must be sufficient to record the highest desired frequencies.

For mining applications, frequencies of approximately 100 Hz are sufficient. Good returns at depths to 500 ft are readily obtainable, and resolution is adequate to provide meaningful interpretation of most mining problems. Broadband data encompassing, perhaps, 80 to 200 Hz have been found to be the most useful for mining applications.

Definition of a given problem is not solely dependent upon resolution of the feature at the ore deposit. The seismic section has valid interpretation data from very near the surface to depths below the ore deposit itself. This is shown in figure 5, wherein a channel sand can be seen affecting the overlying sediments, and faults (fig. 6) can be seen

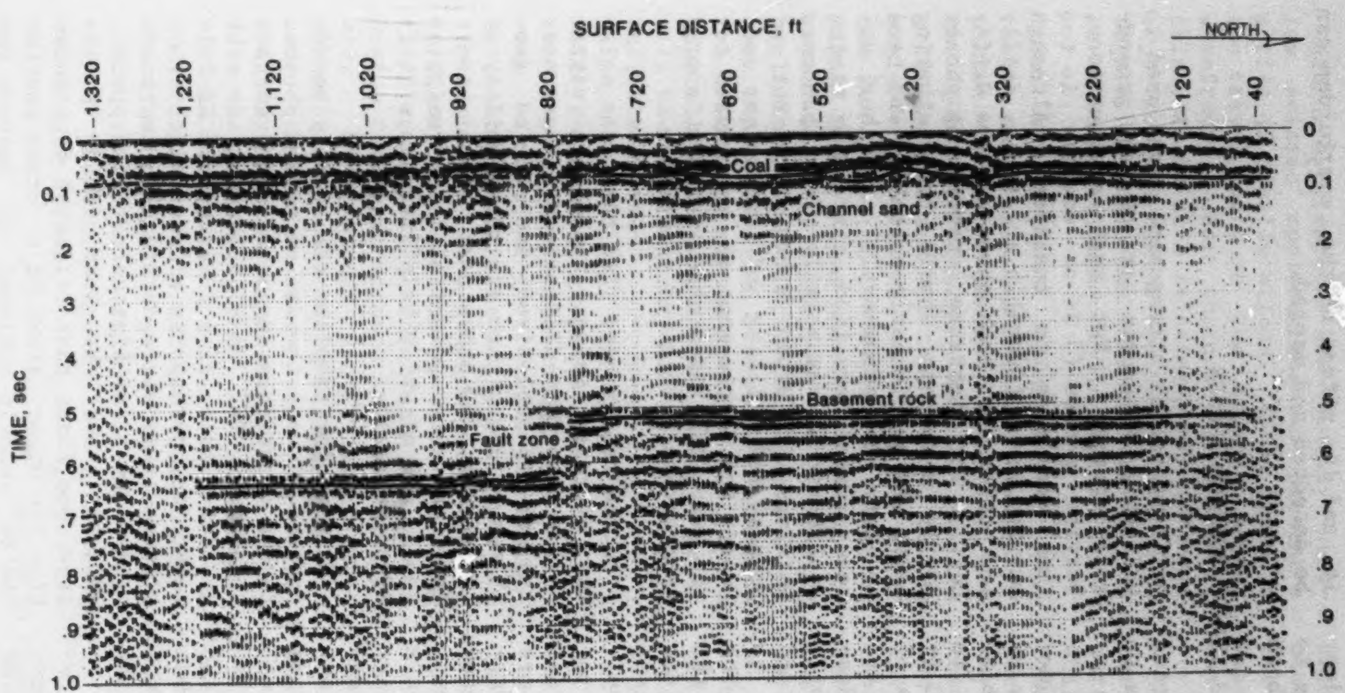


FIGURE 5. - Seismic section showing draping of overlying sediments above channel sand.

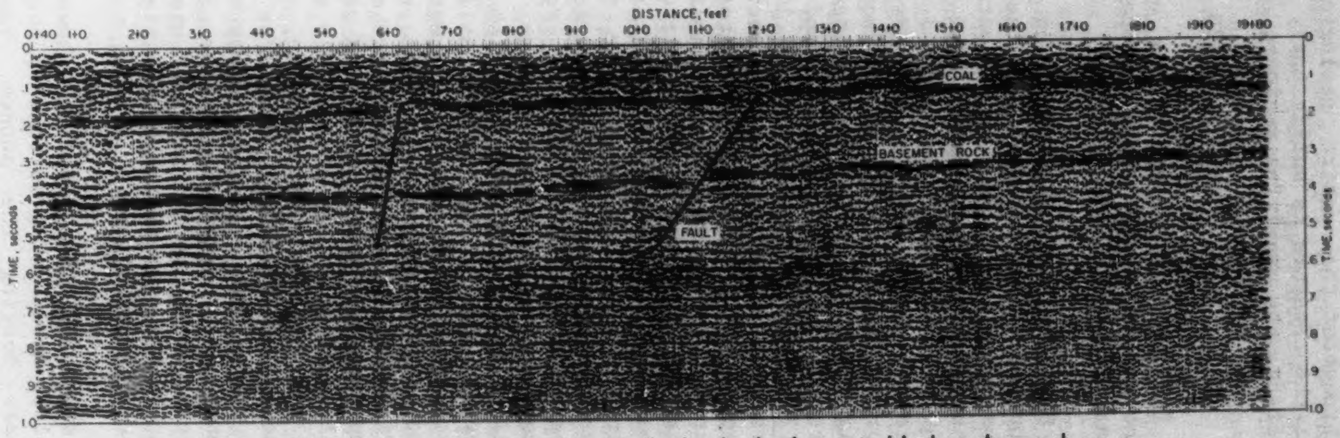


FIGURE 6. - Seismic section showing faults both above and below the coal seam.

throughout the entire geologic section. If the feature is sufficiently large to affect the overlying geology, then both it and its effect on the overburden can be detected and mapped.

BURIED PHONES VERSUS SURFACE PHONES

The question of whether to use buried phones, or merely surface phones, is an important consideration.

Buried phones means putting the geophones in drilled holes at the base of the weathering, or at the water table. These holes could range from 1 ft to 100 ft deep, with depths of 3 to 20 ft probably being nominal. The advantage of buried phones is that high-frequency and high-quality data can be obtained. Hasbrouck (6) and Ziolkowski and Lerwill (13) obtained frequencies of 100 Hz and produced excellent cross sections to depths of several hundred meters. The buried-phone scheme lends itself to the use of hydrophones with their excellent coupling characteristics in water. Also, single phones are used rather than subarrays; hence, phasing differences between phones that invariably result in smearing of the high-frequency data are eliminated. The result is sharp frequency definition. Additionally, good production techniques can be worked out, wherein each geophone position becomes a drilled hole for energy-source explosive emplacement. Hence, each hole has a useful dual purpose, and little additional time is lost in drilling, planting phones, and shooting, etc. For critical or troublesome areas, where one wishes to obtain high detailing in the data, the use of buried phones may be a must. One should be prepared to use buried phones for critical areas.

Bureau of Mines research, however, revolves about using surface phones, sometimes burying the phones just below the surface if it is windy. Occasionally, subarrays, with six phones in a series-parallel arrangement, are used to half-wave attenuate part of the ground-roll energy. For such applications, burying phones (3 ft or deeper) is too time consuming to consider. Although burying seems favorable at times, drilling holes for each phone can be difficult. For instance, retrieving phones may be a concern should the weathering layer be particularly unstable and tend to bury them permanently. Further, Bureau researchers find that many mining applications, especially for abandoned workings in urban areas, are out of bounds for drilling and explosives use. Hence, the development of procedures for the use of repeatable surface energy sources has been given preference.

Bureau researchers find that the seismic approach using repeatable signal sources has many distinct advantages over the use of explosives and buried geophones. Among them are repeatability of the source, the subsequent signal stacking, and the potential for weighted shots, all of which help increase signal-to-noise ratios.

Many mining problems can readily be addressed without the use of buried phones. Bureau experience suggests using surface techniques for most surveys, or the greater portion of any survey, to be followed up, perhaps in a few select troublesome areas where detailed answers are needed, by a few lines of buried phones. Once an answer has been found in one area, one finds that selective extrapolation to other areas of a cross section can be readily achieved to improve the total subsurface analysis.

SINGLE GEOPHONES VERSUS SUBARRAYS

Good field procedures necessitate versatility in using trade-offs to achieve "optimum" results. An important one is the use of single geophones versus subarrays. Both have a value, depending upon the problem being studied and on other circumstances, usually the seismic characteristics existing down to the target area.

Mining applications are usually "shallow seismic" applications and present a series of considerations. Among these is the travel path of the seismic energy wavefront. For shallow applications, this path is at a relatively obtuse angle (Θ_2) from the signal source, as shown in figure 7; while for deeper horizons, it is at an acute angle (Θ_1).

For single-phone applications, this merely means that the seismic energy impinges upon the phone at an angle, and the subsequent energy transfer to the coil is governed by the cosine of the angle from vertical, as shown in figure 8. This, of course, means a loss of signal energy that may be overcome by repeat shots (stacking) or by increasing amplifier gains. Frequency response at the amplifier will, however, be exactly the frequency of the moving coil, which hopefully, if the geophone plant is good, will be the frequency of the incoming reflected wave.

If an array of geophones is used, either to increase total energy returns for weak signals, or to provide ground-roll attenuation, then the angle of the upcoming signal may be significant. As shown in figure 9, the first phone in the array (A') receives the wavefront energy and begins to output signals to the amplifiers. Microseconds later, the second phone (B') receives its signal and outputs signal to the same amplifier, slightly out of phase with the signal from the first geophone. This continues through the entire string of geophones and, of course, is accentuated the further the phones are separated from each other, or if the terrain is hilly. This effect is frequently called phasing or smearing. If high-resolution returns are being sought, this objective will be compromised. For many applications, this may be satisfactory. For others, the important high-resolution signal nuances may be lost forever.

Additionally, manufacturing tolerances for standard geophones are such that some variations in sensitivity occur (7). While it should be understood that these tolerances are quite sufficient for the predominant oil exploration market, they may sometimes be marginal for high-resolution mining seismic applications. The effect of this difference in sensitivity is similar to that described above, in that if the units are in an array, then phasing and frequency smearing can occur. Sometimes it may be a problem--sometimes not.

On the other hand, geophone subarrays are a powerful tool for attenuating unwanted ground-roll signals. If well configured, subarrays may attenuate ground roll by as much as 24 db. Farr (5) describes this procedure. Because ground-roll returns invariably come in during the time window of wanted shallow reflection signals, their attenuation to any degree is helpful. A strong combination can be achieved through repeatable surface sources (single or weighted), subarrays, and carefully chosen offset distances from source to geophones. The trade-off, however, will be some loss of resolution.

In the Bureau's ongoing work, the present emphasis is to develop procedures around single-phone configurations with repeat signal inputs, and other windowing procedures to eliminate ground-roll effects.

AMPLIFIER CHANNELS

Although the simplest of systems, having only one or two channels, can provide useful information for some problem applications, the availability of at least 12 channels is desirable; 24, 48, 96, and up are excellent. However, the more channels, the higher the initial costs, and, possibly, the production costs per mile of coverage. Also, when large numbers of channels are used in linear surveys, offset distances to the far phones become a serious problem for shallow-mining applications. Bureau experience for linear surveys indicates that a good tradeoff for a digital system is 24 channels. For 3-D (three-dimensional)

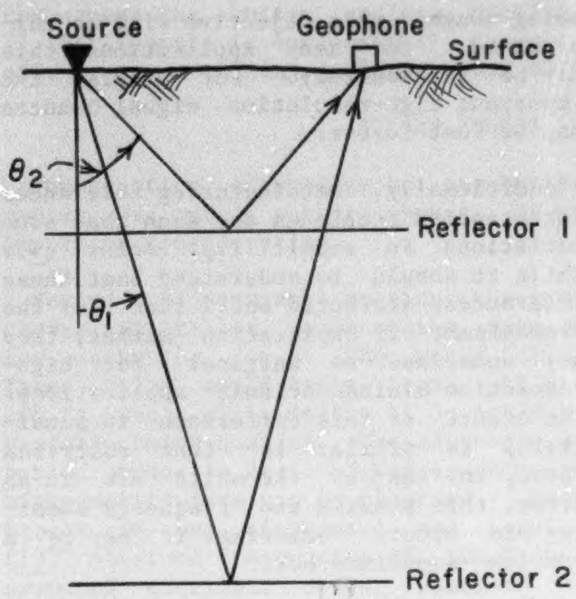

Source
Geophone
Surface
θ_2
Reflector 1
θ_1
Reflector 2

FIGURE 7. - Seismic travel path considerations.

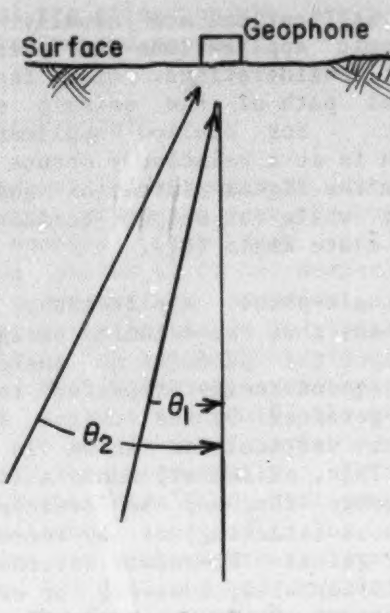

Surface
Geophone
θ_1
θ_2

FIGURE 8. - Geophone response as a function of ray path angle from vertical.

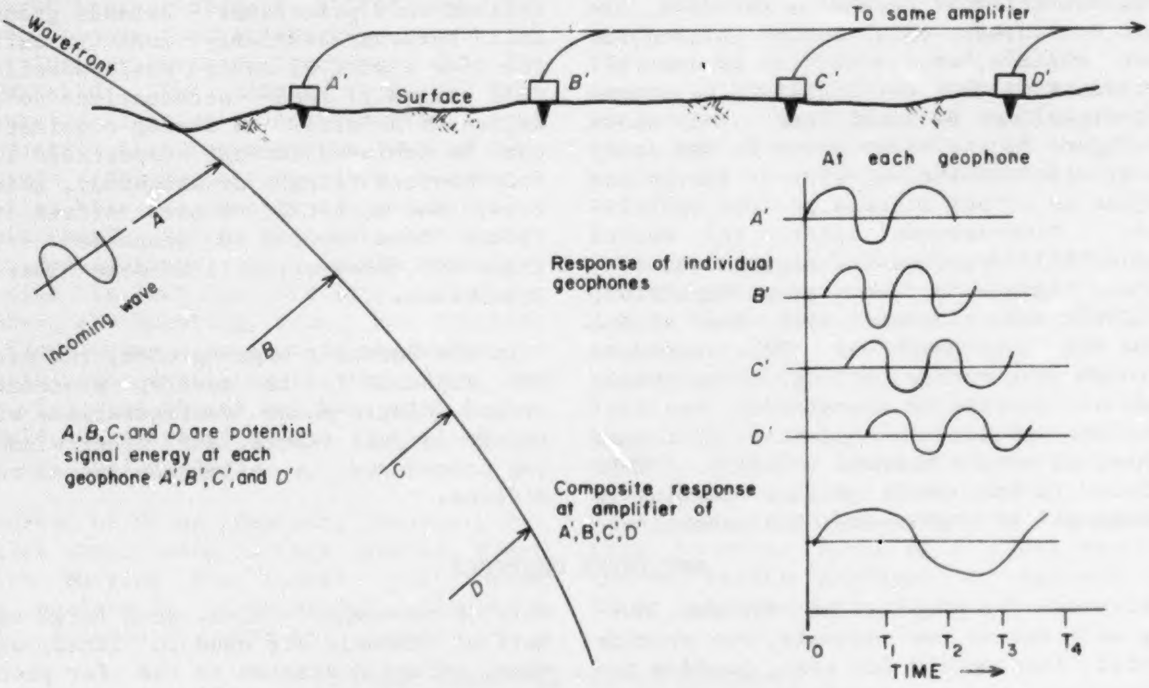

Wavefront
To same amplifier
A'
B'
C'
D'
Surface
At each geophone
Response of individual geophones
A
Incoming wave
B
C
D
A,B,C, and D are potential signal energy at each geophone A',B',C', and D'
Composite response at amplifier of A',B',C',D'
T_0
T_1
T_2
T_3
T_4
TIME →

FIGURE 9. - Effect of wavefront impinging on a subarray.

applications, 48 to 96 channels or more are best.

A 24-channel system allows for up to 12-fold CDP (common depth point) stacking in a linear array, and more if 3-D grid shooting is performed. A 12-channel system providing 6-fold CDP stacking has been found to be minimum for mining applications. The 24-channel system, on the other hand, provides a greater versatility of operation. Triaxial arrangements can be configured, or single versus multiple phones used, without losing too much lineal progress of the line. Also, 3-D grid shooting, which is showing promise of being a powerful technique, can be implemented more readily.

In all of this, however, one should be governed by the magnitude or complexity of the problem being studied. If a small project, such as locating a channel sand washout in front of active mine workings, is being tackled, a 12-channel or smaller system would be adequate. If it is desired to map an area of 1 square mile, it would be better to use a 24-channel or larger system.

Some of the early work by Bureau researchers was with an 8-channel system. It was found to be quite sufficient for many applications, but limiting, at times, for performing large-scale or multiphone studies. Figures 10 and 11 show some of this 8-channel work. The quality is reasonable, but additional coverage, with about the same amount of effort, could have been obtained with a larger system.

INITIAL TESTS--NOISE SPREADS

Of paramount importance in any survey is the need to perform a series of initial tests to seismically characterize the site. This should minimally include an uphole survey in the site area so that the representative velocity distribution can be determined early in the survey to aid site characterization. Secondly, the area should be thoroughly evaluated, on a selective basis, to determine which of a variety of seismic approaches should be used.

The velocity survey is important, because seismic systems essentially measure time of wavefront travel. Hence, in order to determine depths, which is what one really wants, it is necessary to know the wave velocity of the travel path. The ultimate accuracy of any survey will depend upon how well the velocity distribution is known. For much work, a good "average" velocity distribution will yield many solutions. This average velocity can be obtained from an uphole survey in a nearby drill hole, or from a refraction survey over the area, or from computer analysis of moveout-travel path differences. Of these, the velocity uphole survey is the best, although it represents only one region in the survey area. The others provide broad coverage but are heavily statistics dependent. Procedures for conducting these surveys, and the subsequent analysis, are thoroughly covered in the literature, for example Dix (2) and Embree (4).

Initial tests are so important that Bureau researchers unequivocally adhere to the dictum, "Do not proceed with the production survey until you have thoroughly evaluated the site and its optimum field procedure requirement." The sought-for goal is the best possible signal-to-noise ratio in the depth zone of interest. The bane of shallow seismic work is the fact that low-frequency ground roll predominates in the time window of the desired reflections. Its effect is to swamp the amplifiers, and their effective dynamic range, before wanted signals can be recorded at reasonable signal-to-noise levels. Mere stacking of repeat shots, or filtering after the fact at the computer center, will not help. In the first case, the ground roll will be coherent for repeat shots; and weak signals embedded in large groundroll excursions usually cannot be effectively extracted from the data.

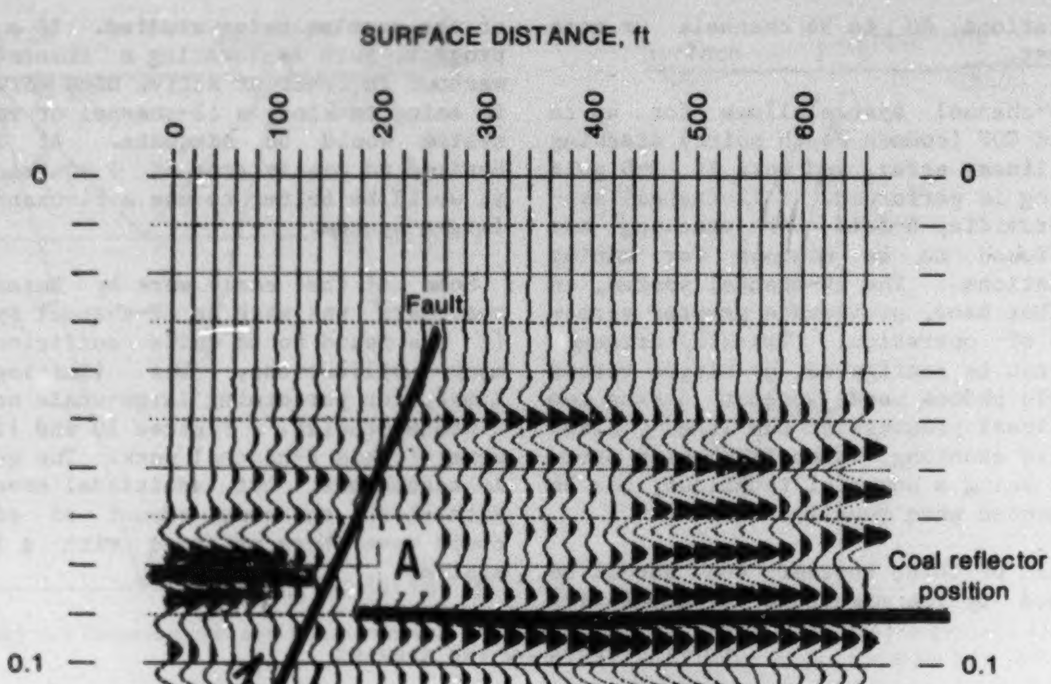


FIGURE 10. - Seismic profile of a coal measure, Colorado.

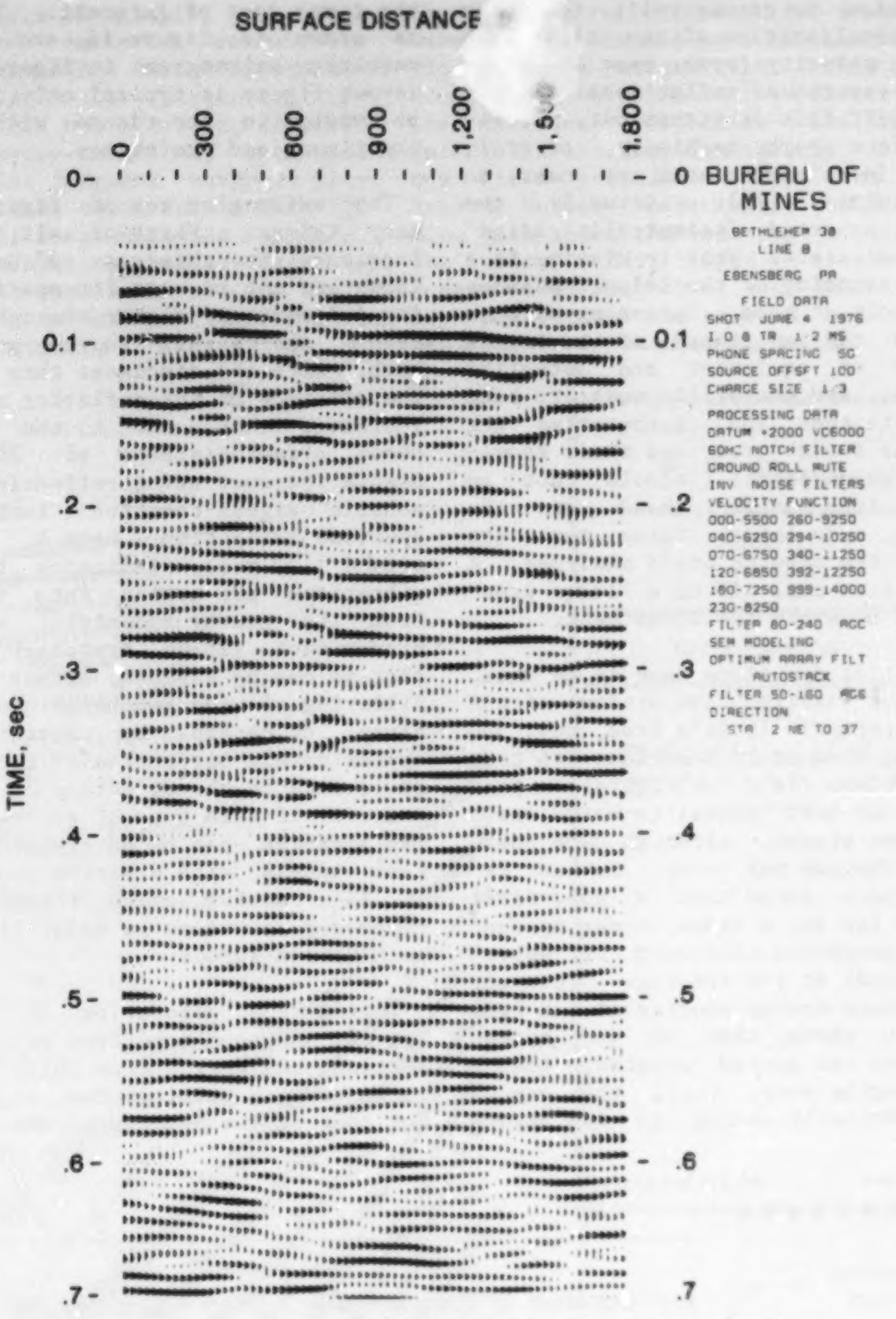


FIGURE 11. - Seismic profile of a coal measure, Pennsylvania.

In addition to ground roll, there may be other peculiarities of the site, such as a high-velocity layer near the surface, or spurious reflections from a nearby cliff face or stream cut, or vibrations from nearby machinery, or traffic (if in an urban area or near an active mining site). Actually, the sometimes reported "seismically blind" area seldom exists. What is missing is a full understanding of the seismic nuances and the tools at hand. Bureau experience has shown the importance of having a repertoire of equipment and potential approaches, and the willingness to take the time to test and characterize the area. The field crew, and their knowledgeable geophysicist, should not be pressured into forging ahead into the production mode before these tests are made and the results field analyzed. A week, or even two, if on a large prospect, will be worth the investment.

Some initial decisions have to be made. What is the desired time window of the zone of interest? If it's from, say, 300 to 500 ft, then it is important to test for the optimum field configuration that will give the best signal-to-noise ratio in that time window. Although each operator's technique may vary, Bureau researchers have found that a good first step is to lay out a close order spread, with the geophones clustered (if subarrays are used) at 1-m spacing. Then one should produce energy sources at 1 m from the nearest phone, then at one spread length, then two spread lengths, then N spread lengths away, until the ground roll is obviously coming in far beyond the time zone of interest. This layout is shown in figure 12, and a set of resulting seismograms in figure 13. The layout figure is typical only, and should be varied in accordance with personal experience and preference.

The seismogram set of figure 13 shows many things. First of all, groundroll frequency (in this case 27 Hz) is now known and can be used for spatial filtering judgments of either the subarrays (if used), the signal source pattern, or both. Farr (5) discusses this procedure. The presence of the reflector at 175 msec indicates that a shot to the first geophone offset-distance of 200 ft will assure its continuing reflection at reasonable signal-to-noise levels. The spurious reflection, band A, coming in around 150 msec, indicates that noise vibrations are coming into the spread from the nearby industrial area. Its predominant 125-Hz frequency indicates that it can be subdued, either by spatial filtering of the geophones in the subarray, or perhaps by pattern shooting approximately perpendicular to the line, or ideally in the direction of the noise source. If this type of source is fixed, its location can be determined from the test spread with a little geometry. If it is a moving source, then it becomes necessary sometimes to delay firing until such noises subdue.

During the generation of the test spread, one should perform selective gain setting changes. From this, a typical range of gain settings can be determined for most of the ensuing survey.

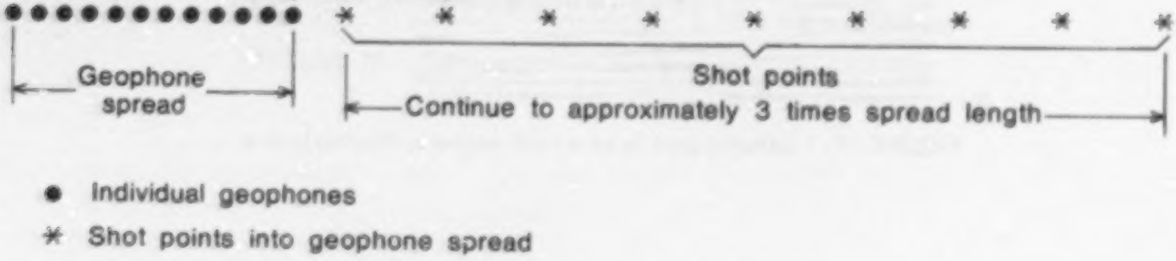

0.5m
2m
4m
Geophone spread
Shot points
Continue to approximately 3 times spread length
● Individual geophones
* Shot points into geophone spread

FIGURE 12. - A noise spread layout.

From the figure 13 example, which was in a location declared "seismically blind," some reasonable first approximations of field layout parameters resulted. Had this not occurred, it would have been necessary to reshoot each shot point with a best judgment weighted pattern configuration, using Farr's work (5) and ground-roll frequencies as a guide. If a pattern shooting had additionally been done, the resulting section would aid in defining an optimum field procedure. In retrospect, the direction of the spurious reflection at 150 msec could have been determined by a concentric circle geophone pattern (1). This layout would have established the direction of the spurious reflection and further influenced the decision towards an optimum field procedure. From the first breaks, Bureau researchers could determine much about the velocity layers near the surface. This helped in establishing judgments on whether or not sharp velocity discontinuities existed and an approximation of weathering depths.

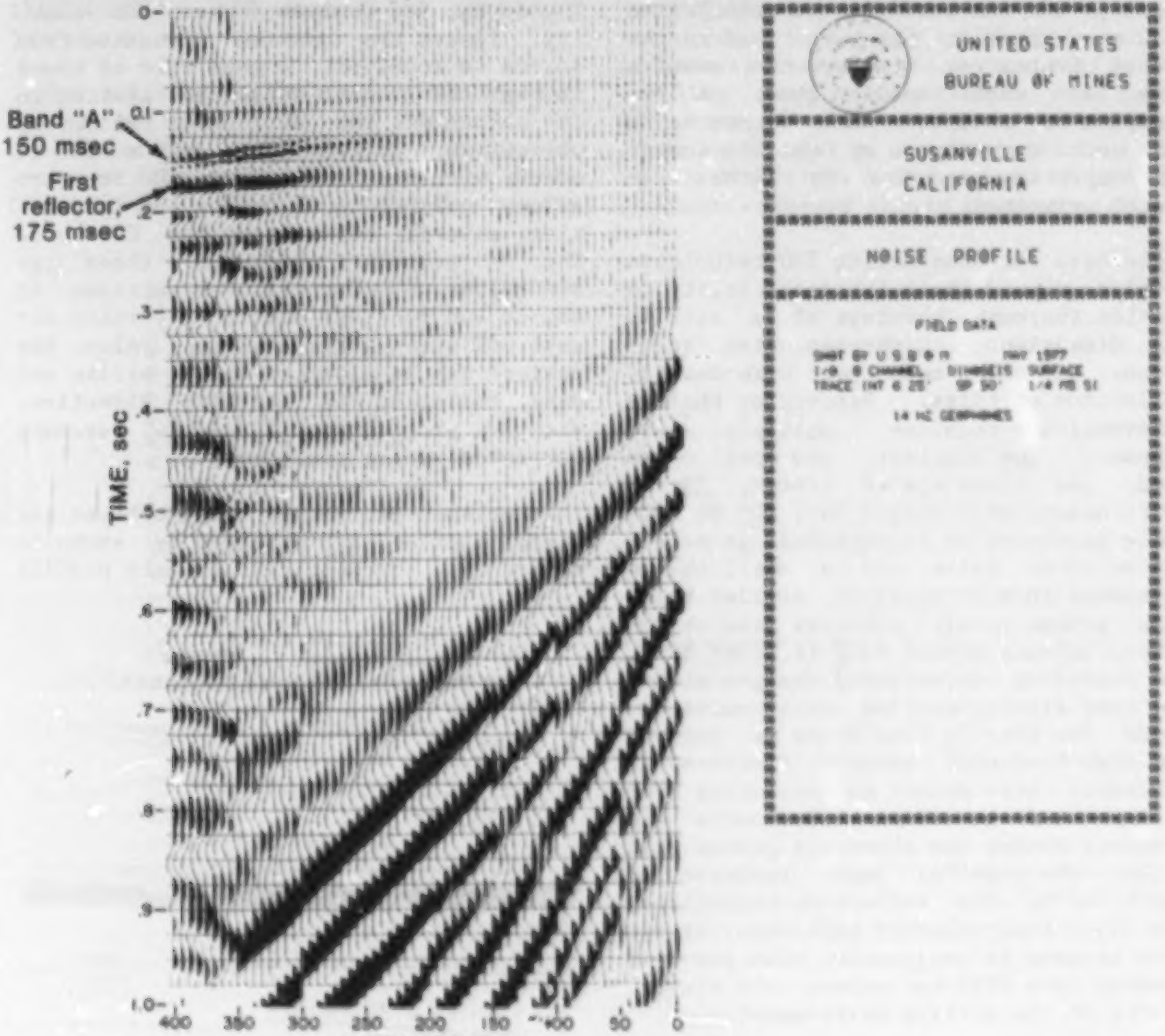


FIGURE 13. - A noise spread profile.

At this point, one will want to decide what further procedures to try to improve signal-to-noise conditions. This always must remain No. 1 in one's mind. It is also important that reflections must be seen in the data, even if only cursory at this point, before signal enhancement procedures, such as Common Depth Point (CDP) or signal averaging (stacking), will aid the quest. If reflections can be seen through the procedures outlined above, one can feel confident that CDP and stacking will help. If they can't be seen, then one should reassess offset, geophone, or shop pattern layout dimensions, or perhaps even the signal source.

3-D SHOOTING TECHNIQUES

Many examples of the effectiveness of 3-D techniques are to be found in oil exploration literature. Some work has been performed for mining application in Europe, and increasingly in the United States. Where it has been used in the United States for mining, the results have been exceptionally good. A few examples of this work are presented in this section; some are by seismic industry companies, and some by Bureau research personnel.

The case for considering 3-D techniques revolves around their inherent ability to provide thorough coverage of a site in all dimensions. High-resolution techniques may often mean just high-density collection of noise. Recovering shallow information requires multiple data schemes. The simplest, and most often used, has close-spaced traces, 20 to 25 ft apart, with single 40-, 60- or 100-cycle geophones or hydrophones in water-filled 10-ft holes, and a small charge detonated in a 1- to 10-ft drilled hole. This scheme rarely achieves the objectives, because ground roll is broad band, and exploding concentrated charges within the poor elastic shallow environment exceeds the elastic limits and is poor in the high-frequency range. A successful procedure must select an operating frequency band in which (1) source and receiver arrays can attenuate ground roll within the physical limit rendered by depth or by dip reflection objectives, and (2) a high-velocity explosion or device is used in conjunction with low-unit loading that will not exceed the elastic limits of the surface environment.

Further, to fulfill the array-attenuation procedures, source and receiver patterns will need to be minimally 40 to 10 ft or longer. The physical requirements of ground-roll suppression are difficult, or perhaps impossible, with conventional profiling procedures, if the objective is shallow or dipping. The depth, and to some degree the velocity, limits the maximum distances from source to receiver. A good rule of thumb is that the longest offset be limited to the depth of the shallowest reflecting objective. Therefore, to map a 500-ft event with stations spaced 100 ft apart allows only 10 to 12 traces of information, severely restricting the CDP fold. The 3-D procedure overcomes these restrictions. Using the 3-D matrices of source and receiver stations permits almost unlimited array lengths. Also, the pattern can be oriented along strike and data collected in the dip direction. With 3-D procedures, 12, 24, 96, or more CDP folds can be recorded.

In seismic profiling, the geophones and signal sources are arranged as shown in figure 14. The resultant seismic profile

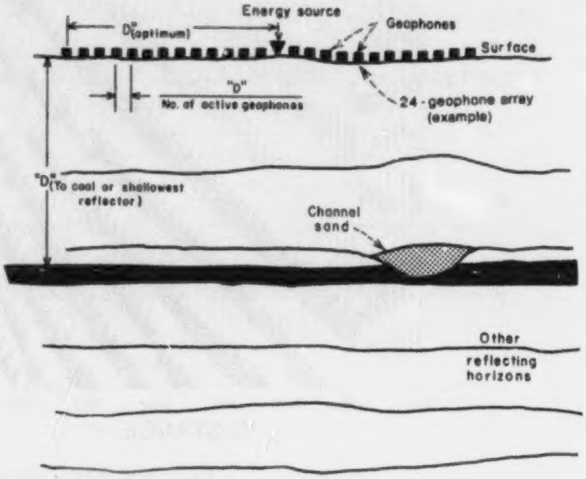
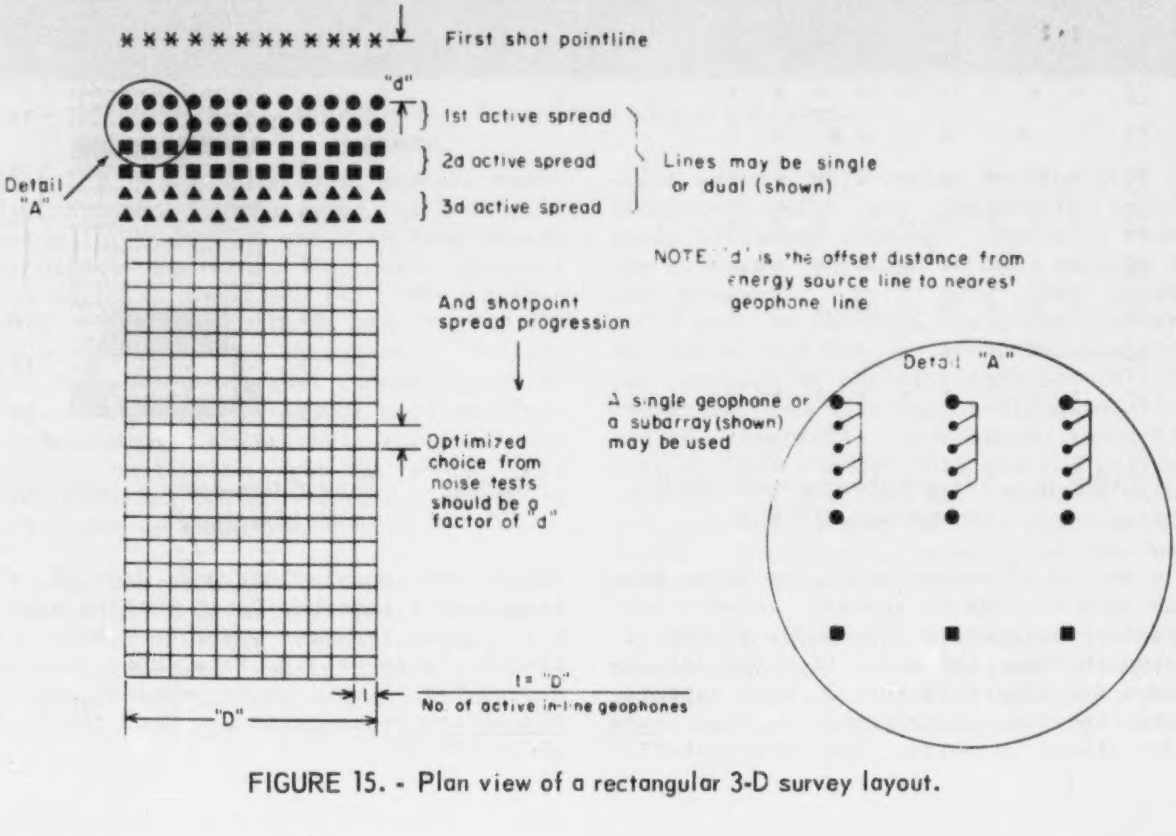

Energy source
Geophones
D° (optimum)
Surface
"D" / No. of active geophones
24-geophone array (example)
"D" (To coal or shallowest reflector)
Channel sand
Other reflecting horizons

FIGURE 14. - Depth of shallowest reflector versus maximum distance to furthest geophone.

will be as shown in figures 4, 5, 6, or 10. These profiles, though valuable, represent only a single cross section of the coal geology.

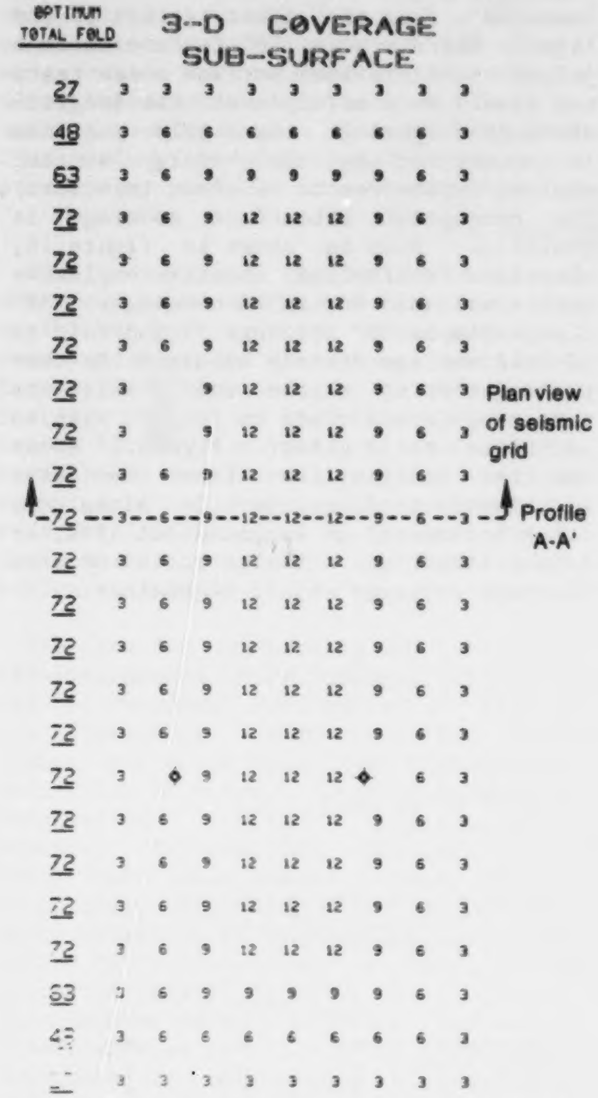
A 3-D survey, on the other hand, provides a multitude of profiles, and in addition, a series of time-depth slices that enables one to examine the ore deposit geology layer by layer. Layout configurations are limited only by the geophysicist's imagination and surface constraints, such as hills, bodies of water, or cultural development. All of these can be effectively surveyed with 3-D procedures.

Where conditions permit, a rectangular layout, such as shown in figure 15, is cost effective. The active spreads are progressively advanced over the grid. Energy inputs are generated on an overlapping grid that starts at a distance "d" from the first active spread line. The distance "d" is chosen from judgments of previous surface noise tests and should be a multiple of the longitudinal grid spacing. Each grid position is energized by the energy source, whether explosives or surface impactors. The consequent subsurface coverage is prolific. This is shown in figure 16, wherein a controlled chamber explosive source was used for rapid coverage. Although simple CDP coverage from 3-fold to 12-fold was immediately obtained by computer gathering of the data, this was increased from 27-fold to 72-fold with no additional field effort. Figure 17 shows one line profile taken from the data. Additional profiles may be along any other horizontal or longitudinal line, or in any direction. The potential of this thorough coverage should be obvious.


First shot pointline
"d"
1st active spread
2d active spread
3d active spread
Lines may be single or dual (shown)
Detail "A"
And shotpoint spread progression
Optimized choice from noise tests should be a factor of "d"
Detail "A"
A single geophone or a subarray (shown) may be used
I = "D" / No. of active in-line geophones
I
"D"

NOTE: 'd' is the offset distance from energy source line to nearest geophone line

FIGURE 15. - Plan view of a rectangular 3-D survey layout.


3-D COVERAGE
SUB-SURFACE
Plan view of seismic grid
Profile A-A'

OPTIMUM TOTAL FOLD									
27	3	3	3	3	3	3	3	3	3
48	3	6	6	6	6	6	6	6	3
63	3	6	9	9	9	9	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3		9	12	12	12	9	6	3
72	3		9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	◆	9	12	12	12	◆	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
72	3	6	9	12	12	12	9	6	3
63	3	6	9	9	9	9	9	6	3
[illegible]	3	6	6	6	6	6	6	6	3
[illegible]	3	3	3	3	3	3	3	3	3

FIGURE 16. - 3-D grid showing immediate fold and optimum total fold.

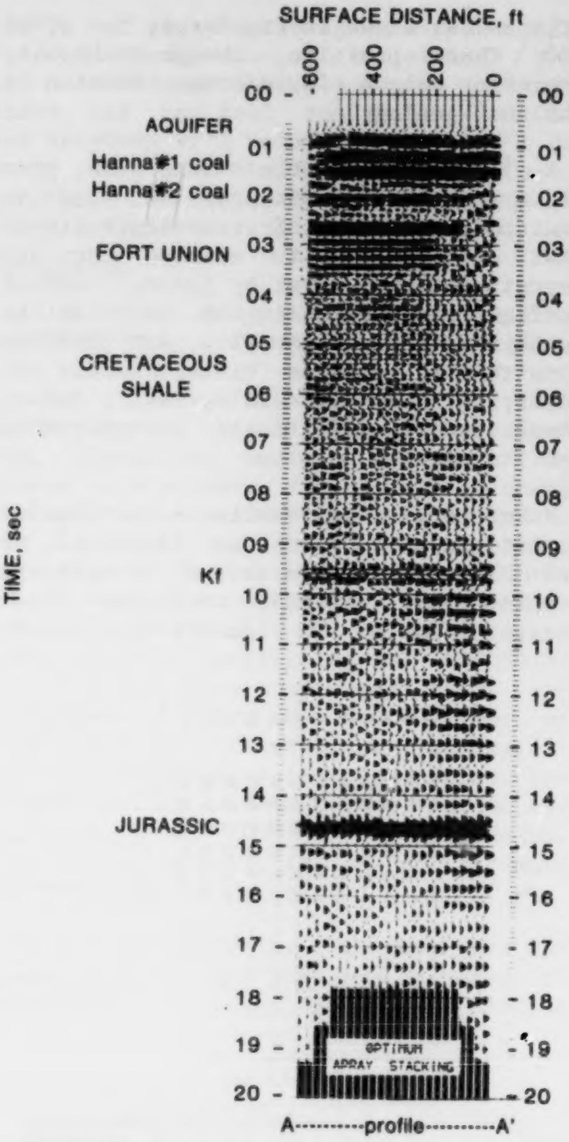


FIGURE 17. - Typical time-depth slice through center of project.

A method of presentation for these data has been evolved by several seismic data processors, wherein time-depth slices are taken through the zone of interest, as shown in figure 18 for a coal deposit. Here the top of the coal is black, and coal itself is white. The interpretation shows the undulations and dip of the seam, and a possible fault cutting across it. These features would have been difficult to ascertain from the profiles alone, but become quite evident on the time-slice presentation. Coal depth is about 270 ft.

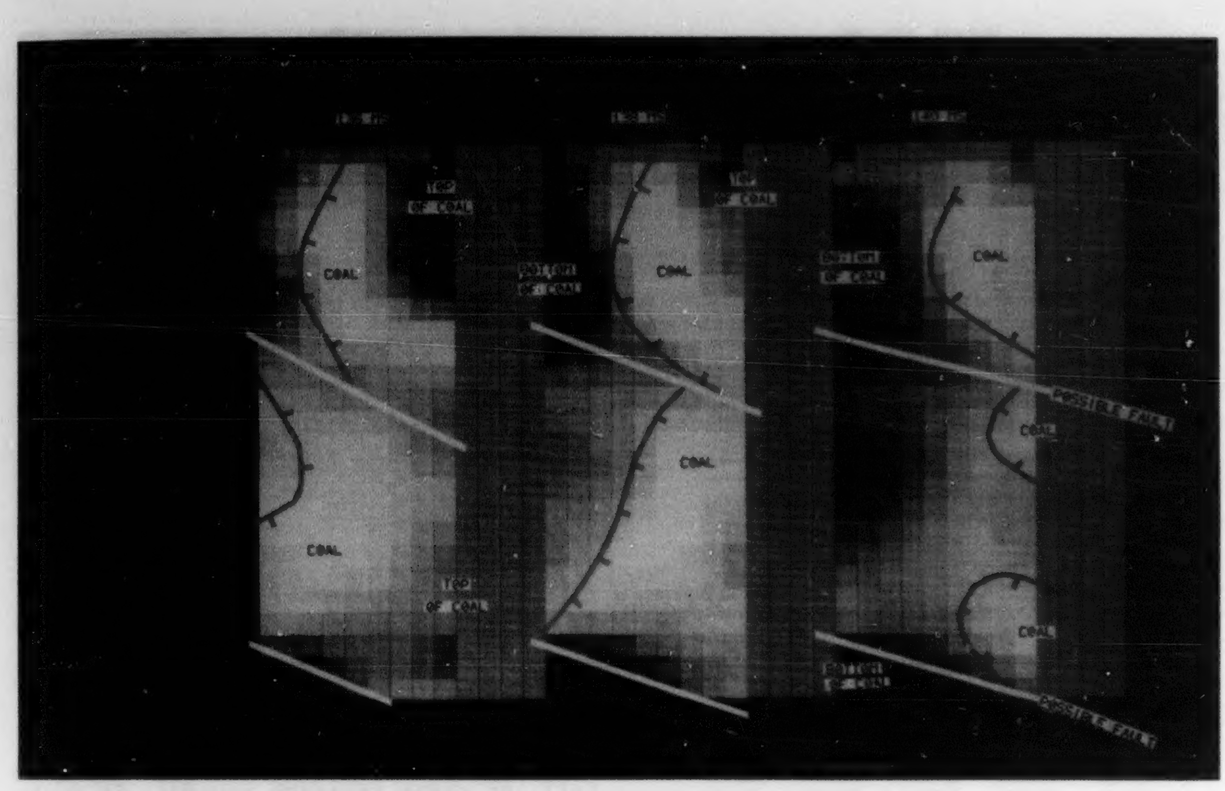


FIGURE 18. - Typical time slice of 3-D seismic.

Figures 19 and 20 provide another example of this procedure using an 8-channel system for recording and a surface impactor signal source for a coal deposit. Coal depth is about 180 ft. Of significance in this example is the high quality of seismic data throughout the section from above the coal to over 2,000 ft deep. Although no abrupt geologic features occur in this coal section, its dipping subtleties can be seen. Had there been any abrupt change, they would have shown up readily.

In another area in southeast Kentucky, a modified 3-D survey was conducted. Because of terrain complexities, the seismic procedure was essentially "fan shooting," but it provided a 3-D coverage over much of the area. The coverage shown is in a hill-valley setting shot in midwinter, which was impassable except along bulldozed roadway. Figure 21 shows the coverage obtained in one section of the prospect. Note here that access could only be obtained along the roadway, as identified by the shotpoint numbers, yet coverage of the site included all of the areas shown in the reflection point map of figure 21. By taking selected data gathers through this covered region, a series of profiles was obtained along the road and in many of the inaccessible areas. From these, the top of the coal seam could be mapped.

Although the 3-D procedure is more costly than profile techniques alone, it provides excellent total coverage data that cannot be obtained by any other

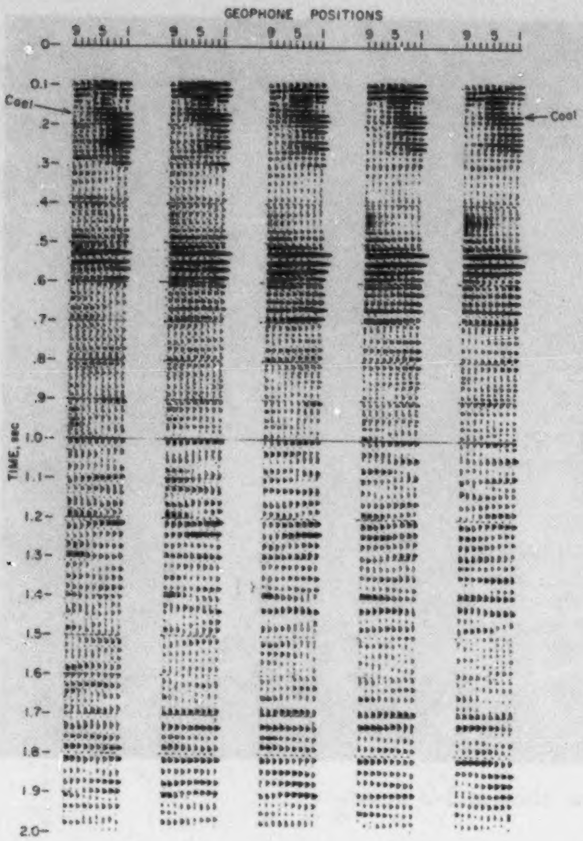
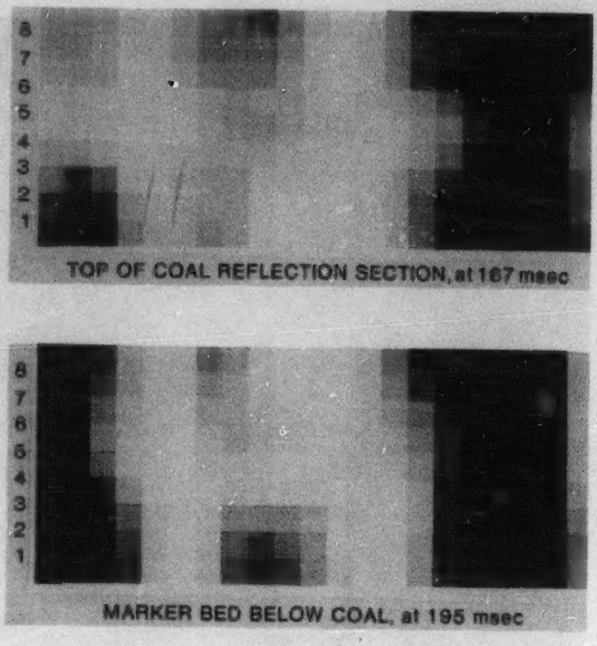


FIGURE 19. - Seismic 3-D profiles using an 8-channel recording system.



TOP OF COAL REFLECTION SECTION, at 167 msec

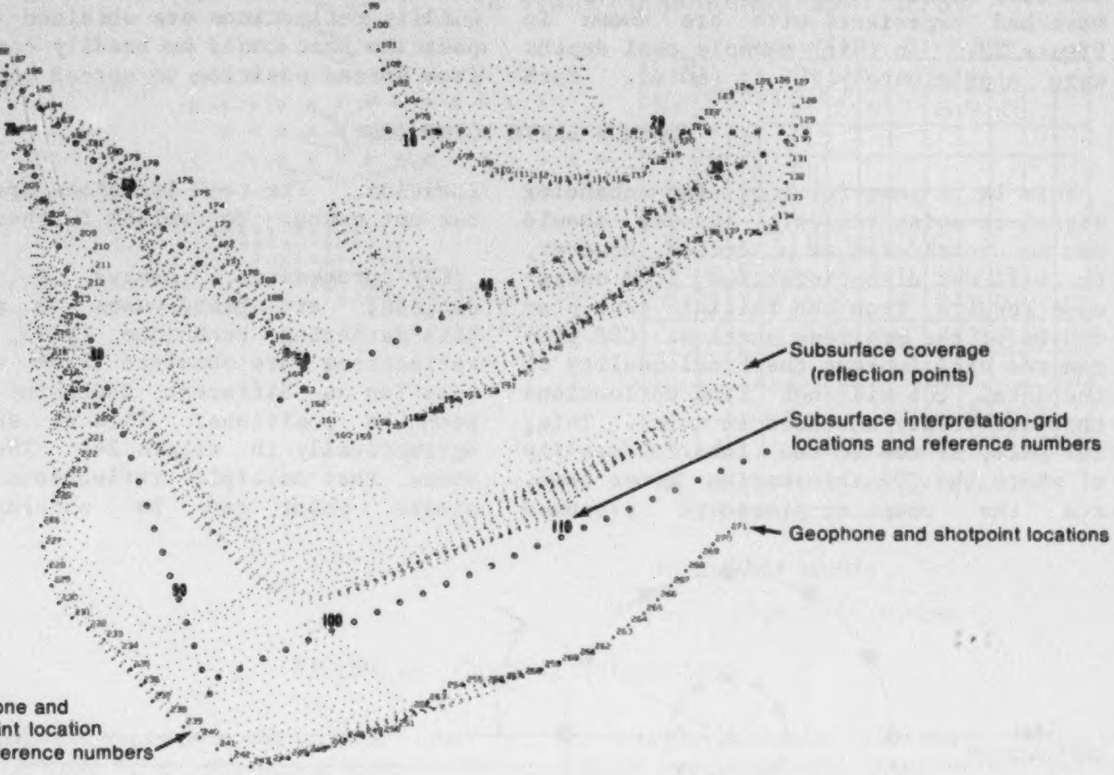
MARKER BED BELOW COAL, at 195 msec

FIGURE 20. - Time slice from 3-D survey.

means. It is recommended for difficult areas where profiling alone seems to fail, or where thorough coverage is wanted to accurately define ore deposit conditions. The trade-off is an investment in excellent data versus merely sufficient data to outline a potential problem area.

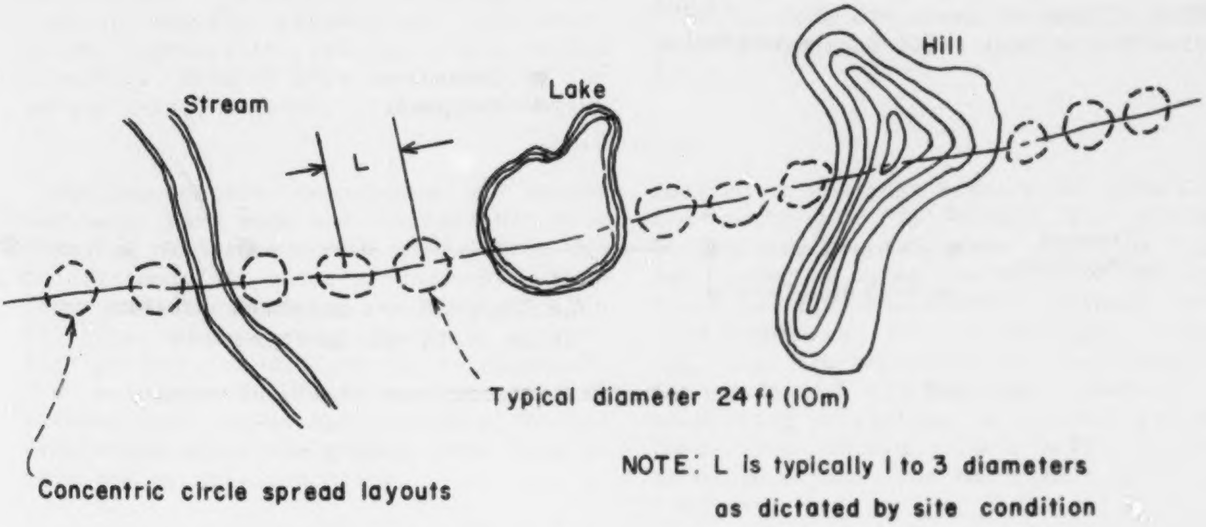
CONCENTRIC CIRCLE TECHNIQUES

At times, there are problem areas that require special attention. This may be because of high-dip conditions, a near-surface high-velocity layer, an adjacent large fault, steep hills, valleys, cliff faces, or cultural noise. A powerful procedure for tackling these areas is a concentric-circle technique developed by several oil companies, and documented by Brasel (1). The power of the concentric circle technique is its ability to act as a directional antenna, from which one may readily determine the direction of spurious noises, and hence, separate them from the wanted reflections. The spreads are relatively easy to lay out and can run as a single continuous trace for line-profiling across complex terrain. Consequently, it is effective in areas where neither normal spreads nor 3-D techniques can be used. This is shown hypothetically in figure 22. Here it is presumed that most of the area is inaccessible for 3-D or continuous profiling. If the concentric circle spreads are laid out wherever possible, a reasonable continuous profile, having good-quality data, can be obtained.


Subsurface coverage (reflection points)
Subsurface interpretation-grid locations and reference numbers
Geophone and shotpoint locations
Geophone and shotpoint location and reference numbers
10
20
30
40
50
60
70
80
90
100
110

Note: Subsurface interpretation—gride locations.

FIGURE 21. - Seismic 3-D coverage in hilly country, Kentucky.


Stream
Lake
Hill
L
Typical diameter 24 ft (10m)
Concentric circle spread layouts

NOTE: L is typically 1 to 3 diameters as dictated by site condition

FIGURE 22. - Concentric circle traverse layout in difficult terrain.

Layout dimensions are site dependent, but some layouts that Bureau researchers have had experience with are shown in figure 23. In this example coal depths were approximately 200 ft (60 m). Each geophone position is a separate trace; hence, CDP stacking could be performed. Quality reflections are obtained at each position that could be readily correlated from spread position to spread position.

COMMON DEPTH POINT (CDP)

This is a powerful tool for enhancing signal-to-noise ratios. Its use should not be considered as a crutch, however, to bail out dissatisfaction from unwelcome results from the initial test procedure of the previous section. CDP procedures will improve the final quality of the data, but will not find reflections that are deeply embedded in noise. This, in part, is due to the inherent reality of where the CDP information comes from, and the computer-procedure presumed location. The two locations are close, but not exact. To explain further:

CDP procedures, always a computer process, are based upon a powerful data-gathering technique (8-9), wherein reflections are obtained from the same location at different shotpoint versus geophone positions. This is shown diagramatically in figure 24. The figure shows that multiple reflections from a single point can be obtained from

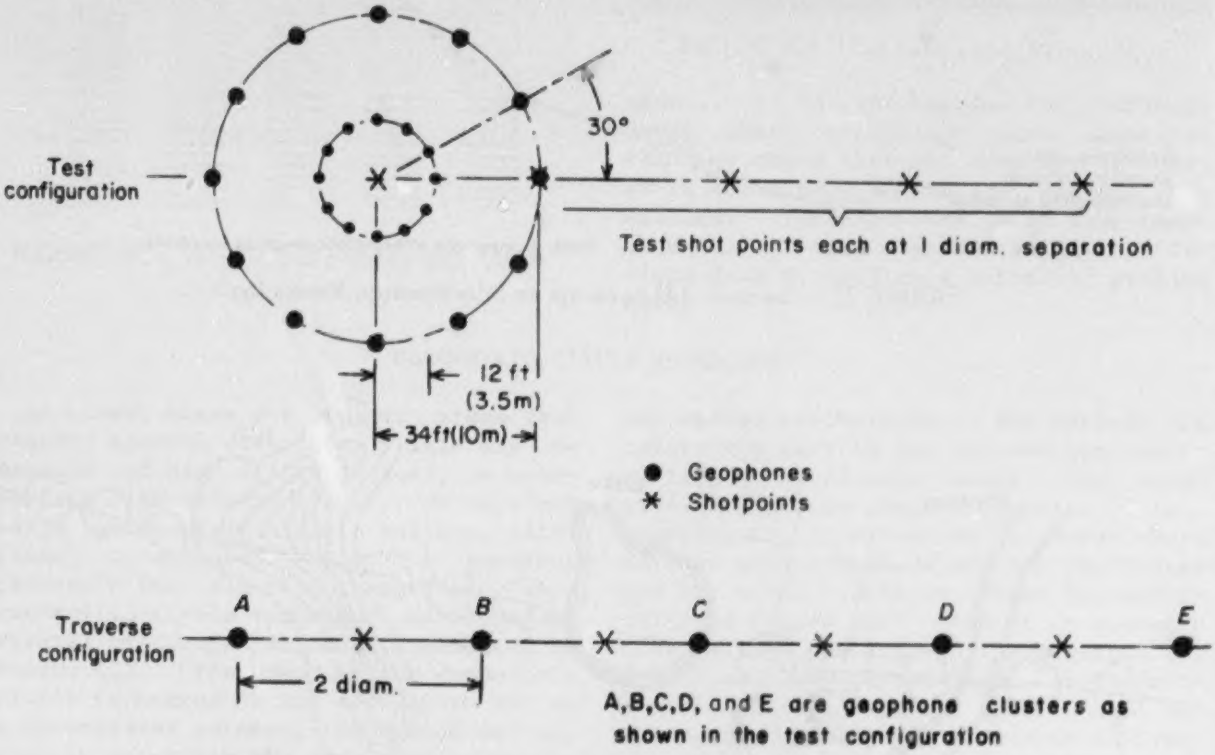
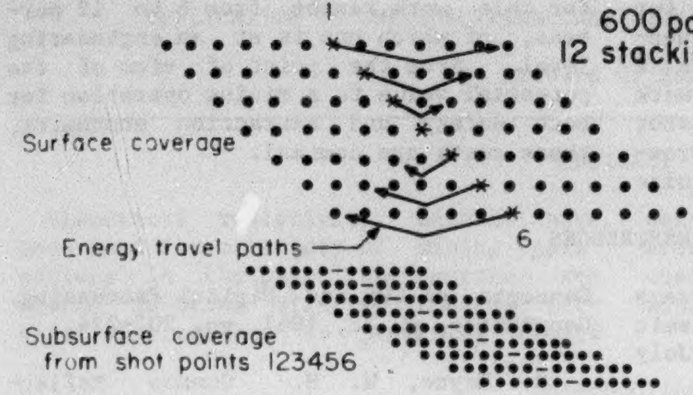

Test configuration
30°
Test shot points each at 1 diam. separation
12 ft (3.5m)
34ft(10m)
● Geophones
✱ Shotpoints
Traverse configuration
A
B
C
D
E
2 diam.
A,B,C,D, and E are geophone clusters as shown in the test configuration

FIGURE 23. - Typical concentric circle traverse layout for difficult areas.

600pct multiplicity
12 stacking channels-split array



Subsurface coverage

SP No.	Depth points											
	1	2	3	4	5	6	7	8	9	10	11	
1	11	12										
2	9	10	11	12								
3	7	8	9	10	11	12						
4	6	-	7	8	9	10	11	12				
5	4	5	6	-	7	8	9	10	11	12		
6	2	3	4	5	6	-	7	8	9	10	11	12
7		1	2	3	4	5	6	-	7	8	9	10
8				1	2	3	4	5	6	-	7	8
9						1	2	3	4	5	6	-
10								1	2	3	4	5

"–" indicates no suburface coverage obtained

✱ = Shot points

SP No. = Shot point number

FIGURE 24. - Stacking configurations.

geophone-shotpoint combinations when a line is shot in a roll-along mode. When computer summed, these add to signal-to-noise enhancement. The exact position, however, is a statistical variable, dependent upon travel path parameters, such as varying frequencies, angle of reflection, scattering, anistropy, and complex signal propagation considerations. Hence, it can smear frequencies somewhat. None of this discounts its inherent value, however, if used knowingly. In shallow seismic reflection work, one should progressively test-select gathers from a good reflector to see whether the final data are actually enhanced or degraded before arbitrarily gathering the maximum possible combinations.

Bureau of Mines work has shown that a 6- to 12-fold CDP stack is usually sufficient to enhance most shallow reflection data.

CONCLUSIONS

Shallow seismic techniques are useful for many hard rock and sedimentary mine planning purposes. Much useful data can be obtained from these procedures. When they are supplemented with selective drilling, an enhanced picture of subsurface geologic conditions can be obtained. The seismic data fill in the picture between drill holes and provide a clearer understanding of the geology than can be obtained by drilling alone.

Although many of the procedures are still in development, enough has been learned in the past 6 years to show that the technology is viable for mining. Many mining companies are fielding their own crews or using the services of contract seismic companies. Seismic service companies are increasingly offering their new expertise to the industry. The essence of the use remains in conducting sufficient a priori seismic tests to obtain the best signal-to-noise window for the depth zone of interest.

The linear seismic line is useful for many applications. For total comprehensive coverage, the 3-D approach is a must. Commercial costs for linear work will be approximately $100 per shot point, while 3-D coverage will be approximately $40 per grid position. Crew size for this work ranges from 6 to 12 persons, of which one is at an engineering level. From the point of view of the potential value to a mining operation for both safety and extraction economics, these costs are nominal.

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APPLICATION OF THE ELECTRICAL RESISTIVITY METHOD TO MINING PROBLEMS

By Richard G. Burdick[1]

ABSTRACT

Electrical resistivity methods have been used for a variety of mining applications in the past, and current research is being directed towards further applications to premining hazard detection and monitoring solution mining areas. Some of the methods are discussed with examples of past mining applications.

INTRODUCTION

The use of the electrical resistivity methods in mining situations is finding expanding application. They may be used to locate geologic faults, measure the degree of fracturing within in-place rock, locate abandoned mine workings or tunnels, and define the limits of solution fronts used in solution or in situ mining. The method is not a panacea for all mining problems, but, when used on a case-by-case basis, it is a versatile tool for a wide range of mining problems. It has the advantage that it may be used from the surface or underground as well as in drill holes.

The two resistivity methods most commonly used by the Bureau for mining applications are the Wenner method (appendix A) and the pole-dipole method (appendix B) (1, 4, 8, 10). In general, the Wenner method is used to investigate larger volumes of earth materials where its inherent averaging effect can be used to advantage. The pole-dipole method is used where more discrete samplings of the earth are required, particularly at greater depths. This paper describes applications of resistivity methods to the solution of various mining problems by the Bureau.

Discussion of Resistivity Principles

As with other geophysical methods, resistivity does not detect the target-of-interest directly, but rather measures the electrical properties in an area, and from these an inference may be drawn as to the presence or location of the target.

Even the most competent rock contains small voids or microfractures that contain water and various dissolved salts and gases. These solutions allow the passage of electrical current and form the basis for the resistivity measurements. A dense, competent rock with few microfractures will contain relatively less moisture than a more porous rock and will normally exhibit a much higher resistance to the passage of an electrical current than will the more porous rock. Thus, the resistivity methods do not measure solution fronts, abandoned mine workings, etc., directly, but simply measure the changes in the volume resistivity caused by their presence.

The equipment for making these measurements can be as simple as using a battery for injecting current into the ground and a sensitive voltmeter to measure the resulting potential, or as complicated as the Bureau's prototype Automatic Resistivity System, which closely controls the current injected and has an elaborate measurement system that accurately measures to 10 mv (0.00001 volt) and records all data, including measurement locations, on magnetic tape for later computer analysis. The intermediate range equipment includes many commercially available units, as well as somewhat more sensitive equipment designed as prototype devices (5).

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The apparent resistivity of a volume of earth using the Wenner method is calculated by the formula:

$$\rho_{app} = \frac{\Delta V}{I} k,$$

where ρ_{app} = apparent resistivity, in ohm-meters.

I = current injected into the ground, in amperes.

ΔV = the voltage potential between two electrodes resulting from the above current.

k = a geometry factor of $2\pi a$.

a = electrode spacing, in meters.

From this equation it may be seen that when the electrode spacing is kept constant there is a direct relationship between apparent resistivity and the measured $\Delta V/I$. If the earth moisture changes there will be a corresponding change in the measured $\Delta V/I$ ratio.

Use of Resistivity To Predict Caveability of Ore Bodies

As a fracture zone within an ore body below the water table would be expected to contain relatively more moisture than the surrounding ore, it would also be expected to show a lower resistivity than the other ore. Based upon this premise, studies were conducted at two mines using block-caving extraction methods. Four or five separate areas were tested and relative caveability values (best, medium, and poor) were assigned based upon the relative apparent resistivities of the areas. In general, these predicted relative caveability values were borne out when the areas were mined at a later date (Source: Unreported Bureau investigations during 1966-68).

Use of Resistivity To Define the Zone of Fracturing Around a Tunnel

Based upon the general logic that is the greater the degree of fracturing the lower the apparent resistivity, a brief study at one hardrock mine seemed to show the zone of fracturing occurring around a tunnel blasted through the rock. In this case, the Wenner Array sounding method was used as described in appendix A. The electrode spacings were expanded in small increments in order to detect the edge of the fracture zone. From this test, the fracture zone appeared to extend to 5 ft out from the walls of the tunnel (Source: Unreported Bureau experiments).

Use of Resistivity To Locate Geologic Faults

The Bureau has used the resistivity traverse method on several occasions to determine the surface location of faults in mining areas. The principles used in this type investigation are shown in figure 1. The fault may be detected either if it serves as a drain for surface moisture resulting in a somewhat drier zone around the fault, or if it tends to contain more moisture than the surrounding area. If the fault has shown enough vertical displacement to bring dissimilar lithologies near enough to the surface to be within the measurement zone, a third possibility for detection exists. All three phenomena have been observed in the past. The method has been used in two uranium fields in Wyoming and at a hardrock mine in Colorado with good results (9).

Use of Resistivity for Detection of Abandoned Mines in Proximity to Current Mining Activities

Both the Wenner and pole-dipole methods have been used for the detection of abandoned mines. However, the pole-dipole method shows the better resolution for for this purpose, because the Wenner

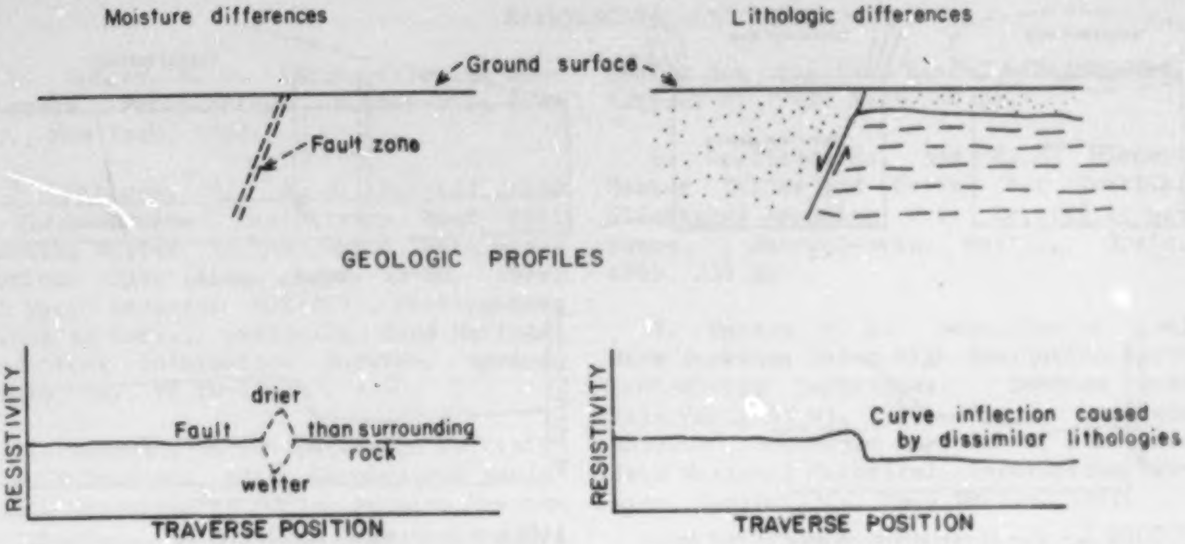
Moisture differences
Lithologic differences
Ground surface
Fault zone
GEOLOGIC PROFILES
RESISTIVITY
drier
Fault
than surrounding rock
wetter
TRAVERSE POSITION
RESISTIVITY
Curve inflection caused by dissimilar lithologies
TRAVERSE POSITION

FIGURE 1. - Use of resistivity to detect faults.

measurement volume is much larger for a given depth of investigation than the pole-dipole and therefore is not as sensitive to detection of a void. This is shown by figures A-2 and B-1. The premise made for this type investigation is that the old workings, whether air- or water-filled, will show a different apparent resistivity than the surrounding earth material. Figure 2 shows the use of the pole-dipole method to detect a void. The lines A, B, and C would be run at the same time with the automated system. From the points where the void was detected on A, B, and C, the void may be located in a manner similar to that shown on the lower right figure. Either the pole-dipole or the Wenner methods have been used in Florida, Illinois, Kentucky, Colorado, Korea, and Israel for locating various underground voids, usually with a high degree of success. Further work is being done in this field to try to increase the ability of the method to detect smaller tunnels at greater depths (7). The Bureau's automated system is still in a testing and modification stage. It is anticipated, however, that the method will be fully operational in about a year and, after that time, a similar system could be constructed by interested parties or by the original contractor for such parties.

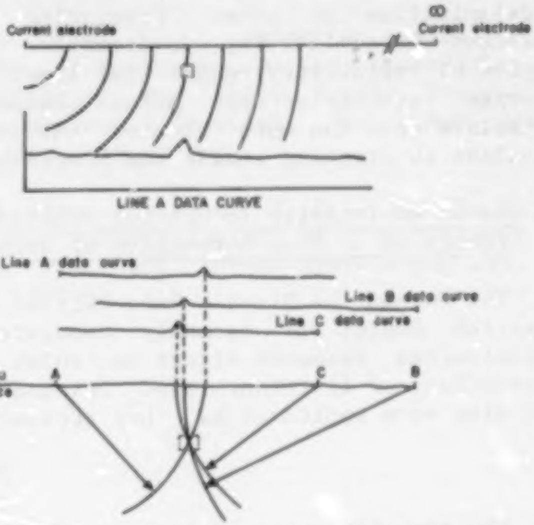
Current electrode
∞
Current electrode
LINE A DATA CURVE
Line A data curve
Line B data curve
Line C data curve
Ground surface
A
C
B

FIGURE 2. - Void detection with pole-dipole method.

Use of Resistivity To Predict Roof Falls

The Bureau has been conducting research for several years, both in-house and under contract (2), to devise an automated roof fall warning device using resistivity principles. The assumption made in this case is that the delamination or fracturing cracks preceding a fall will cause a change in volume apparent resistivity, which can be interpreted as an indication of failure (fig. 3).

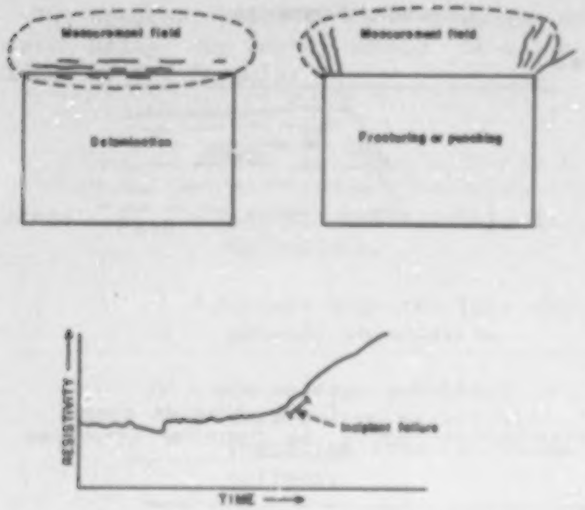
Measurement field
Delamination
Measurement field
Fracturing or punching
RESISTIVITY
Incipient failure
TIME

FIGURE 3. - Use of resistivity to predict roof falls.

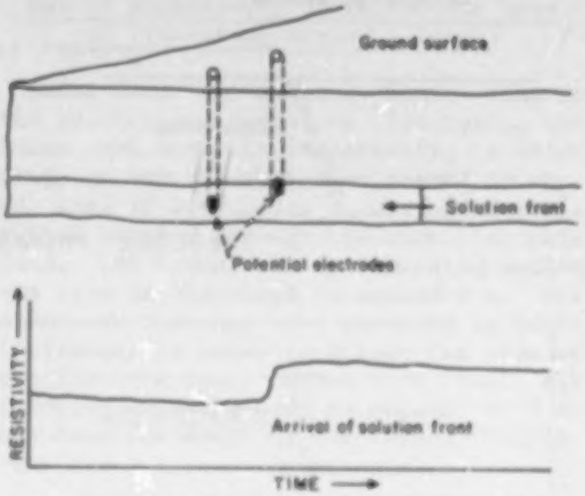
Ground surface
Solution front
Potential electrodes
RESISTIVITY
Arrival of solution front
TIME

FIGURE 4. - Use of resistivity to monitor solution fronts.

This figure shows how cracks caused by delamination or other fracturing can affect the resistivity measurements. The plot of resistivity versus time shows the normal scatter of data points preceding failure and the much greater change in values as cracking starts and progresses.

Use of Resistivity To Monitor Solution Fronts in In Situ Extraction of Ores

The Bureau of Mines' Twin Cities Research Center has recently completed a contracted research effort in which the solutions or lixiviants used for uranium mining were monitored as they approached the edge of the mine area. This method can be used to monitor the progress of solution mining as well as to detect the unwanted migration of the solution beyond the extraction zone. The method relies on the fact that the conductivity of the solution is quite dissimilar from that of the natural moisture in the rock, and when it flows into the measurement field, a change in apparent resistivity will occur. The method has the advantage that the solution movement can be detected in the zone between drill holes rather than just at the drill hole, as had been the case in the past. Figure 4 shows in general how the technique operates (3).

CONCLUSIONS

The described uses of various resistivity methods in a number of mining environments show their versatility for solving a variety of problems. One of the method's strong points is its adaptability for use on the surface or in an underground environment. Another is the simplicity of making measurements and interpreting the resulting data. In many cases, this means that an interpretation of the data may be made while at the test site.

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[2] Items 2, 3, and 7 are available for reference at the Denver Research Center, Bureau of Mines, Denver, Colo.

APPENDIX A.--DESCRIPTION AND INTERPRETATION OF THE WENNER METHOD

The Wenner method uses a configuration of four equally spaced electrodes, as shown in figure A-1. The current is injected between electrodes C_1 and C_2, and the potential difference is measured between electrodes P_1 and P_2. Figure A-2 shows the approximate current and potential fields for a Wenner electrode configuration. It is generally assumed that the potential being measured at a given site represents a summation of the apparent resistivities to a depth approximately equal to the electrode spacing and located between the two potential electrodes.

The Wenner method is used in two ways. In the first the electrode spacing is held constant and the entire array is traversed across the ground surface. This results in looking at a large area to a constant depth. This constant-depth traverse may be used to look for geologic faults, shallow voids, lithologic contacts, etc.

The second method is to maintain a constant center to the electrode array and incrementally increase the electrode spacings. This results in a deeper and deeper measurement depth with an associated larger and larger measurement volume. From this, data sounding curves may be constructed for interpretation of the earth materials at depth by use of a method such as shown in figure A-3 or by means of the sounding curves developed by Mooney and Orelleno (6). The volume of earth material expands as the cube of the electrode spacing, which tends to have an averaging effect on irregular subsurfaces and to obscure small features at greater depths.

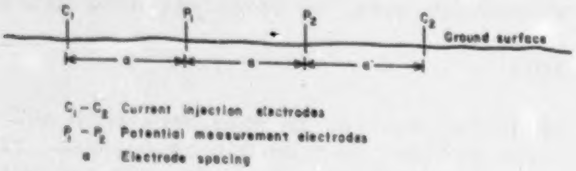

C_1
P_1
P_2
C_2
Ground surface
a
a
a
$C_1 - C_2$ Current injection electrodes
$P_1 - P_2$ Potential measurement electrodes
a Electrode spacing

FIGURE A-1. - Wenner array electrode configuration.

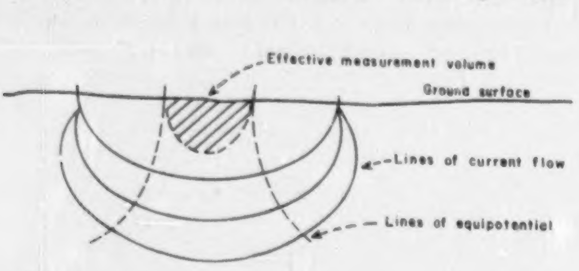


FIGURE A-2. - Approximate current and potential fields for Wenner array.

The averaging effect is useful when trying to determine the properties of large volumes of material, such as measuring the fracture density in an ore body or measuring solution fronts for in situ mining.

The equation for calculating Wenner apparent resistivity is

$$\rho_{app} = \frac{\Delta V}{I} k \text{ apparent resistivity, ohm-meters.}$$

where ΔV = potential difference in volts between P_1 and P_2.

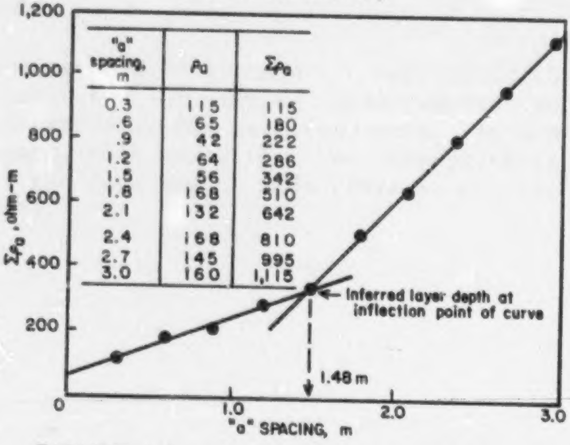
I = current injected between C_1 and C_2 to create potential, in amperes.

k = a geometric constant.

a = electrode spacing, meters.

$k = 2\pi a$ (0.3048 if a is in feet).

($k = 3\pi a$ to a $4\pi a$ for underground measurements.)



"a" spacing, m	ρ_a	$\Sigma\rho_a$
0.3	115	115
.6	65	180
.9	42	222
1.2	64	286
1.5	56	342
1.8	168	510
2.1	132	642
2.4	168	810
2.7	145	995
3.0	160	1,115

FIGURE A-3. - Moore cumulative method of depth interpretation.

APPENDIX B.—DESCRIPTION AND INTERPRETATION OF POLE-DIPOLE METHOD

The pole-dipole electrode configuration used by the Bureau is a modification of the method developed by the English speleologist, Bristow, for detecting caves. It uses several current electrodes, one at a time, near the measurement site and the other at effective infinity (5 to 10 times the largest separation between the other current electrode and the potential electrodes). The potential electrodes are moved as a pair with a constant, predetermined separation between them. Figure B-1 illustrates the electrode configuration used and the electrical fields developed by this method. As may be seen by comparing this figure with figure A-2, the volume of material being measured for a given depth of investigation is much smaller than for the Wenner; this results in a much higher detection resolution for a given sized target at a given depth.

The method may be used for performing either a constant-depth traverse or a depth sounding, but in practice a combination of the two is run, resulting in a cross-sectional view of the earth below the array position. The amount of data resulting from this cross-sectioning is so vast that the interpretation is done by computer. For this reason, a field interpretation of the data is not possible as with the Wenner method. Figure B-2 shows a few of the measurement fields developed during this procedure using the automated resistivity system. As may be seen when comparing the potential fields from numerous current electrode positions, a high degree of data redundancy is created, increasing the accuracy of the method.

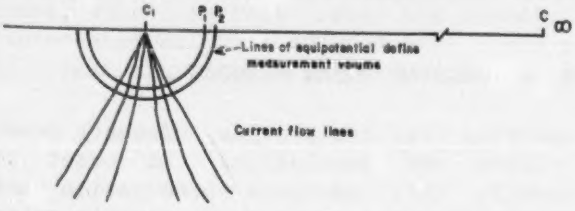


FIGURE B-1. - Approximate current and potential fields for pole-dipole (Bristow) method.

The equation for calculating pole-dipole apparent resistivity is, in ohm-meters,

$$\rho_{app} = \frac{2\pi}{\frac{1}{r_1} - \frac{1}{r_2}} \frac{\Delta V}{I}$$

where r_1 = distance $C_1 - P_1$, in meters.

r_2 = distance $C_1 - P_2$, in meters.

ΔV = voltage potential measured between P_1 and P_2.

I = current injected at C_1 to create potential, in amperes.

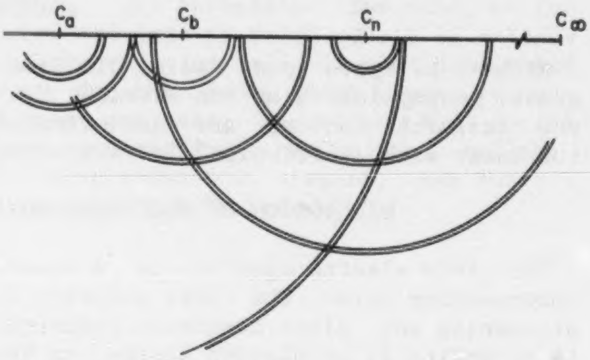


FIGURE B-2. - A few of the measurement fields developed by the pole-dipole method.

ELECTROMAGNETIC GROUND RADAR METHODS

By Richard J. Leckenby[1]

ABSTRACT

The Bureau of Mines has developed a number of electromagnetic ground probing radar techniques for use by the mining industry to detect and map potential mining hazards, such as abandoned mines, water-filled fractures and faults, veins, and well casings. This paper gives insight to some of the techniques and how they are used for detection and mapping of various hazards.

INTRODUCTION

The Bureau of Mines has been actively involved in electromagnetic ground probing radar methods for over 6 years. In that time frame, the research effort has evolved from experiments proving the fundamentals and capabilities of the technique to the development of better detection systems with improved processing and display capabilities that will be useful and practical to the mining industries. The Bureau has actively pursued research and development in a number of electromagnetic methods, each having its own unique advantages and disadvantages.

In general, it is believed that electromagnetic techniques can or will provide the mining industry a tool for rapid and undisruptive detection and mapping of geological features and potential hazards, such as abandoned mines, faults, fractures, veins, and well casings. The intent of this paper is to present an overview of some of the electromagnetic techniques the Bureau has and is developing, and how these techniques are used for the detection and mapping of potential hazards.

ACKNOWLEDGMENTS

Richard L. Myers and James J. Snodgrass, geophysicists at the Bureau's Denver Research Center, are acknowledged for their work as Principal Investigators and Technical Project Officers, for the development of many of the electromagnetic techniques discussed in this paper.

DISCUSSION OF ELECTROMAGNETIC GROUND PROBING RADAR METHODS

The term electromagnetic is a broad, encompassing term. The first priority in discussing any electromagnetic technique is to define it by placing limits on the technique to be used. In the case of the techniques to be discussed here, the methods will be confined to the use of electromagnetic radiation for the measurement of the electrical and magnetic properties of the ground as a hazard detection method. Further restrictions will be placed to include only that band of frequencies in the electromagnetic spectrum that can provide adequate penetration and resolution. In order to provide this adequate penetration and resolution in a variety of minable materials, frequencies between 5 and 500 MHz are normally used. This translates to wavelengths between 60 and 0.6 m in free space, and 20 to 0.2 m in a ground media having a velocity of propagation approximately one-third that of free space. Using this band of frequencies, penetration from 2 to 300 m can be achieved for most rock types, for a large number of rock media where the technique would be useful, penetration of 30 m or better has been or soon will be achievable.

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The underlying principle for electromagnetic techniques is to determine and measure the effects the ground has on the electromagnetic radiation as it passes through the ground. The major physical parameters affecting the propagation are the frequencies of the waves, the complex permittivity, and the complex permeability of the ground. Whenever there is a change in either the complex permittivity or permeability, the electromagnetic wave propagation characteristics will change. That is, reflection or refraction will occur with associated changes in velocity, amplitude, and polarization taking place for a wave of a given frequency. It is these changes that are measured, and with general geological knowledge of an area, an interpretation can normally be made, assigning a given wave propagation change to a probable geological feature.

Although there is a variety of different methods, techniques, and instrumentation associated with the various electromagnetic methods, there are also some commonalities. For instrumentation, figure 1 illustrates the key components found in most of the electromagnetic ground probing radar techniques. Likewise, most of the methods are used in either a reflection (radar) type or a transillumination (one-way) type mode. Figure 2 is illustrative of the two modes, where in this case the anomaly, represented as an abandoned mine, has a different electrical property than the surrounding rock material. This change causes reflection and refraction to occur. The reflection, the echo, or the refracted wave is received via an antenna and then is recorded. It should be noted that for the radar shown where two antennas are used, a surface and an air wave are received, as well as an echo. For single-antenna systems, the surface wave would not be present.

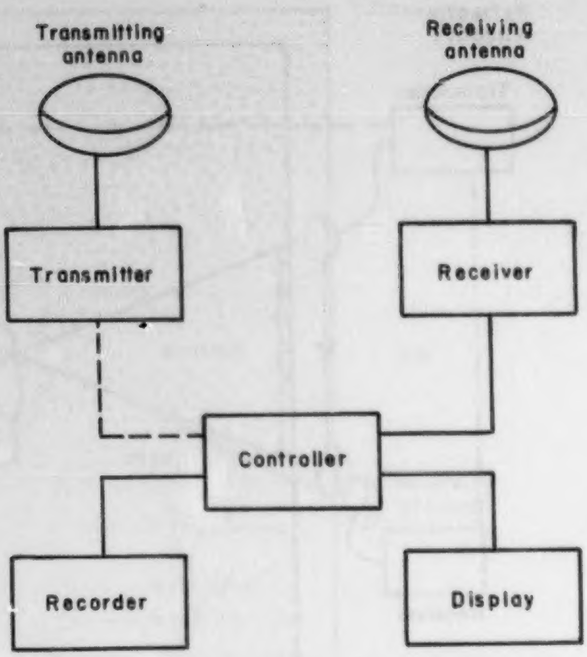

FIGURE 1. - Key components for most ground radar systems. The linking of the transmitter to the controller is optional depending on the technique used.

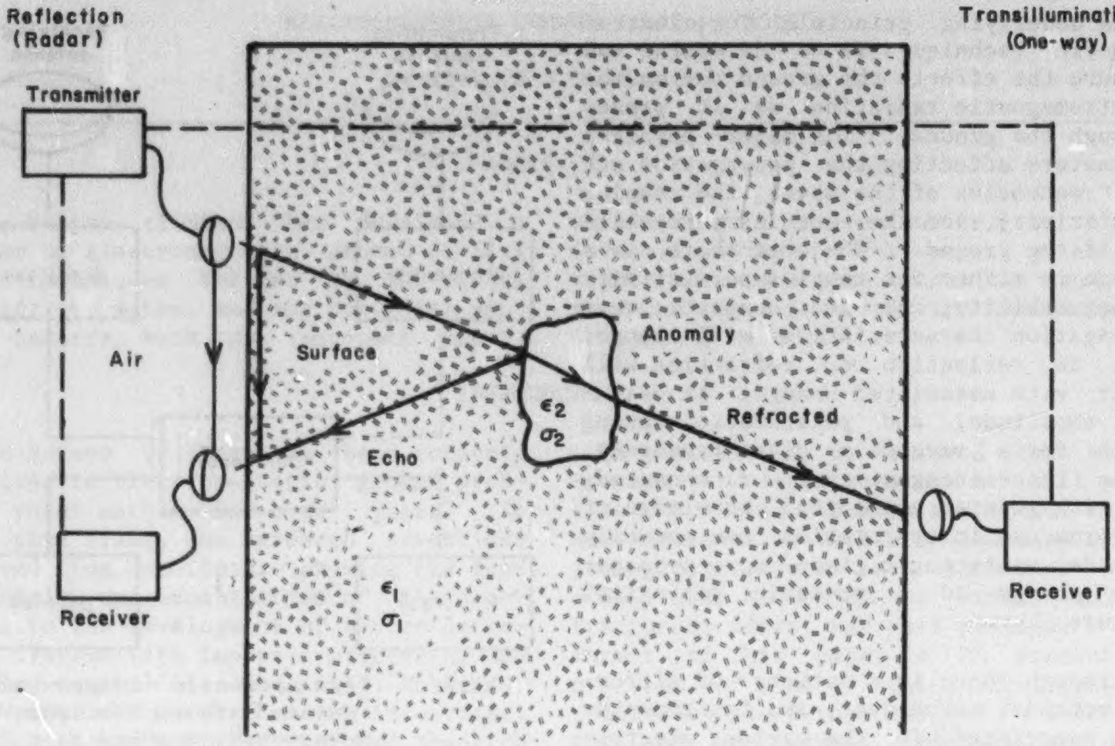

FIGURE 2. - Two modes of operation: reflection and transillumination. The solid black lines with the arrows indicate named travel paths of electromagnetic waves from the transmitter to receivers. ε_1 and σ_1 are the dielectric constant and conductivity of the rock media. The dielectric constant and conductivity of the anomaly are represented by ε_2 and σ_2. The link between the transmitter and receiver is optional depending on the technique used.

Some of the electromagnetic methods that can be used are (1) short-pulse, (2) synthetic-pulse, (3) continuous-wave (CW), (4) FM-CW, (5) chirp, and (6) tone burst. They all have their own advantages and disadvantages, unique electronics, data processing, and display requirements. From this list, the Bureau has mainly been concentrating its efforts toward developing and researching the short-pulse, synthetic-pulse, and continuous-wave methods.

Figure 3 illustrates the short-pulse method. The method can be applied from the ground surface, a borehole, or underground in a mine. The method of detection is usually done by making voltage versus time measurements, using a sampling oscilloscope or high-speed transient digitizer. The measured voltage is then recorded either on digital or analog tape. In the radar mode, the anomaly is detected by receiving the echo of the transmitted wide-frequency band pulse at a delayed time.

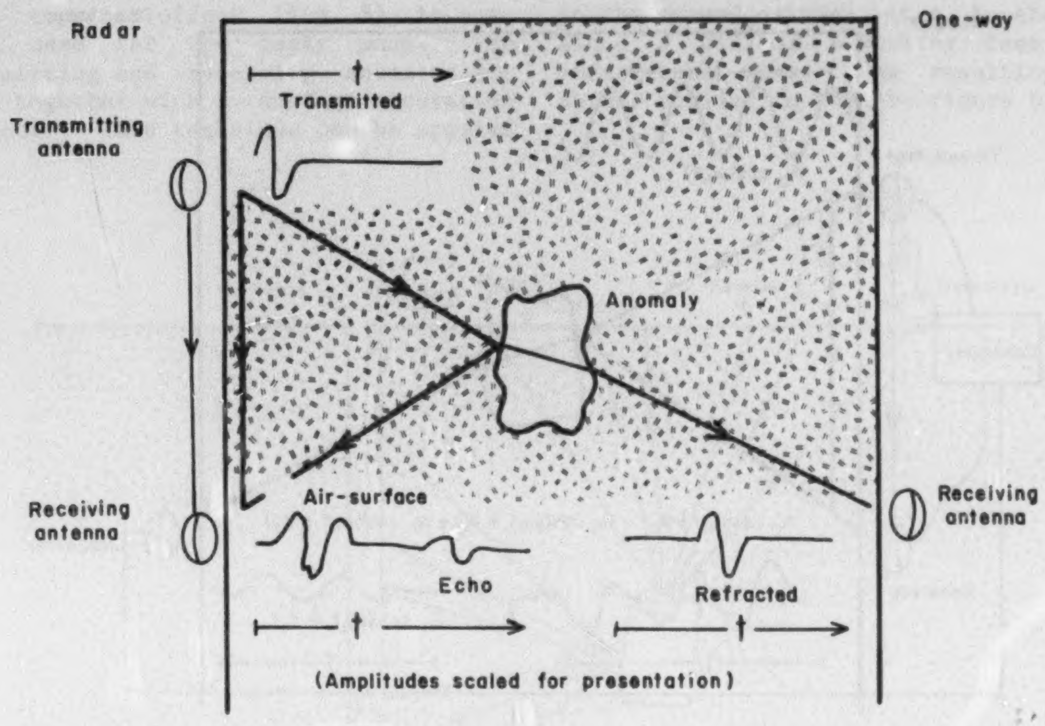

FIGURE 3. - Short pulse radar methods. Representative pulse signals with respect to time, t, of the transmitted and received signals are shown for different travel paths indicated by the arrows for both radar and one-way modes.

For the one-way mode, the signal will arrive at a different time and amplitude than what would be expected if there were no anomaly; thus, an anomaly within the ground can be detected. Some of the advantages of the short-pulse method are that (1) the instrumentation can be assembled using standard off-the-shelf equipment, (2) preliminary results can be quickly displayed, (3) data processing techniques have already been developed for a variety of signal problems, and (4) interpretation of the data is possible without elaborate processing or display techniques.

The continuous-wave method is shown in figure 4. The name is indicative of the technique, in that a continuous wave is transmitted into the ground. The technique is normally used in a transillumination mode, and the Bureau uses the method mainly from cross-borehole surveys. A measurement of the amplitude and phase of the received signal is made and usually referenced to the transmitted signal. Some of the advantages of the CW method are that the electronics can be designed for a given frequency being transmitted, and thus can be made more efficient than those used in the short-pulse method. The recorded data can also lend themselves to tomographic-type displays, making them easier for interpretation. The disadvantage of the technique is that it normally has to be applied in a transillumination mode, because the return signal in the radar mode is so confused that it is impractical to interpret the data.

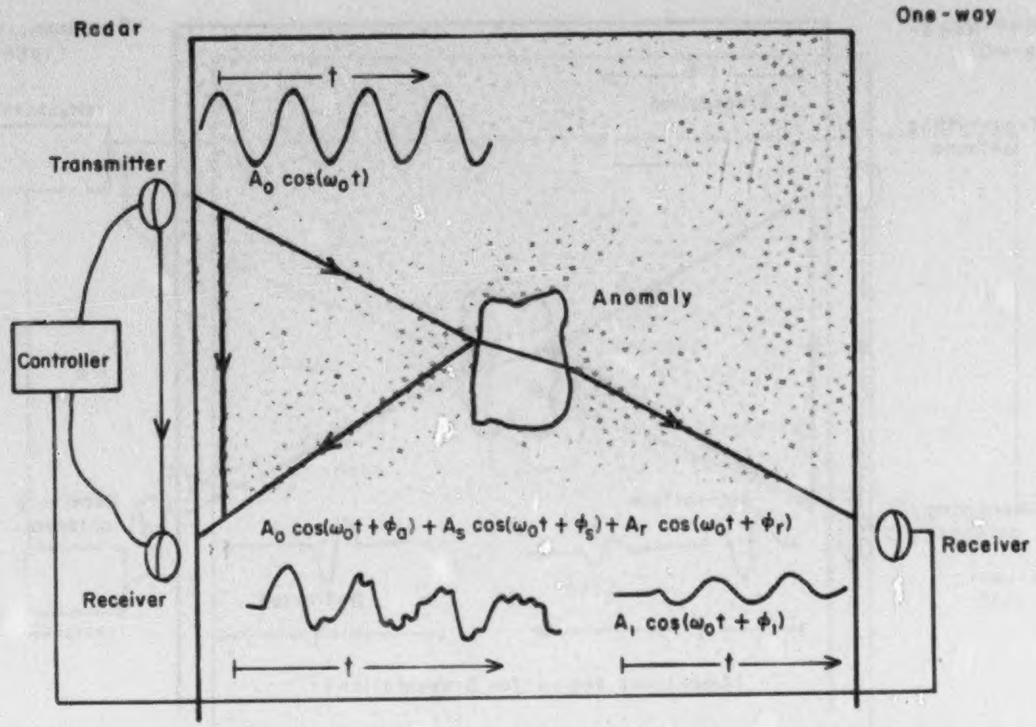
Radar
One - way
t
$A_0 \cos(\omega_0 t)$
Transmitter
Anomaly
Controller
$A_0 \cos(\omega_0 t + \phi_0) + A_s \cos(\omega_0 t + \phi_s) + A_r \cos(\omega_0 t + \phi_r)$
Receiver
Receiver
$A_1 \cos(\omega_0 t + \phi_1)$
t
t

FIGURE 4. - Continuous wave methods. Representative continuous wave signals with respect to time, t, for the radar and one-way mode. A_o, A_s, A_r, and A_1 are amplitudes of the continuous waves with an angular frequency of a_o for the transmitted surface, reflected, and refracted waves, respectively. ϕ_s, ϕ_r, and ϕ are phase constants of the surface, reflected, and refracted waves.

The synthetic-pulse method is an attempt to take advantage of the positive points from the CW and short-pulse methods. The electronics for transmitting and receiving the signals are somewhat similar to that used for continuous waves, but for the synthetic method, hundreds of discrete frequencies are transmitted and recorded. By processing the return signals, pulselike data can be constructed. The synthetic-pulse method can be used in a radar mode, and at this time it shows promise of at least doubling the effective range of the short-pulse method.

Although all of the different methods can be used for detection of the various anomalies, locating or mapping the potential hazards or geological features is necessary in order to make the methods practical. The mapping is performed normally by moving the antenna or antennas relative to the anomaly, and thus causing a change in travel distance for the signals. This movement can be performed by a number of ways. Some of the more common ways are known as (1) constant offset; (2) moveout or common depth; and (3) cross-borehole mapping.

The constant offset (fig. 5) is commonly used for the radar mode. The transmitting and receiving antennas are moved together with a constant separation distance. This technique can be applied on the ground surface, in a borehole, or along a wall or a working face in an underground mine. The resulting data appear similar to that in figure 6.

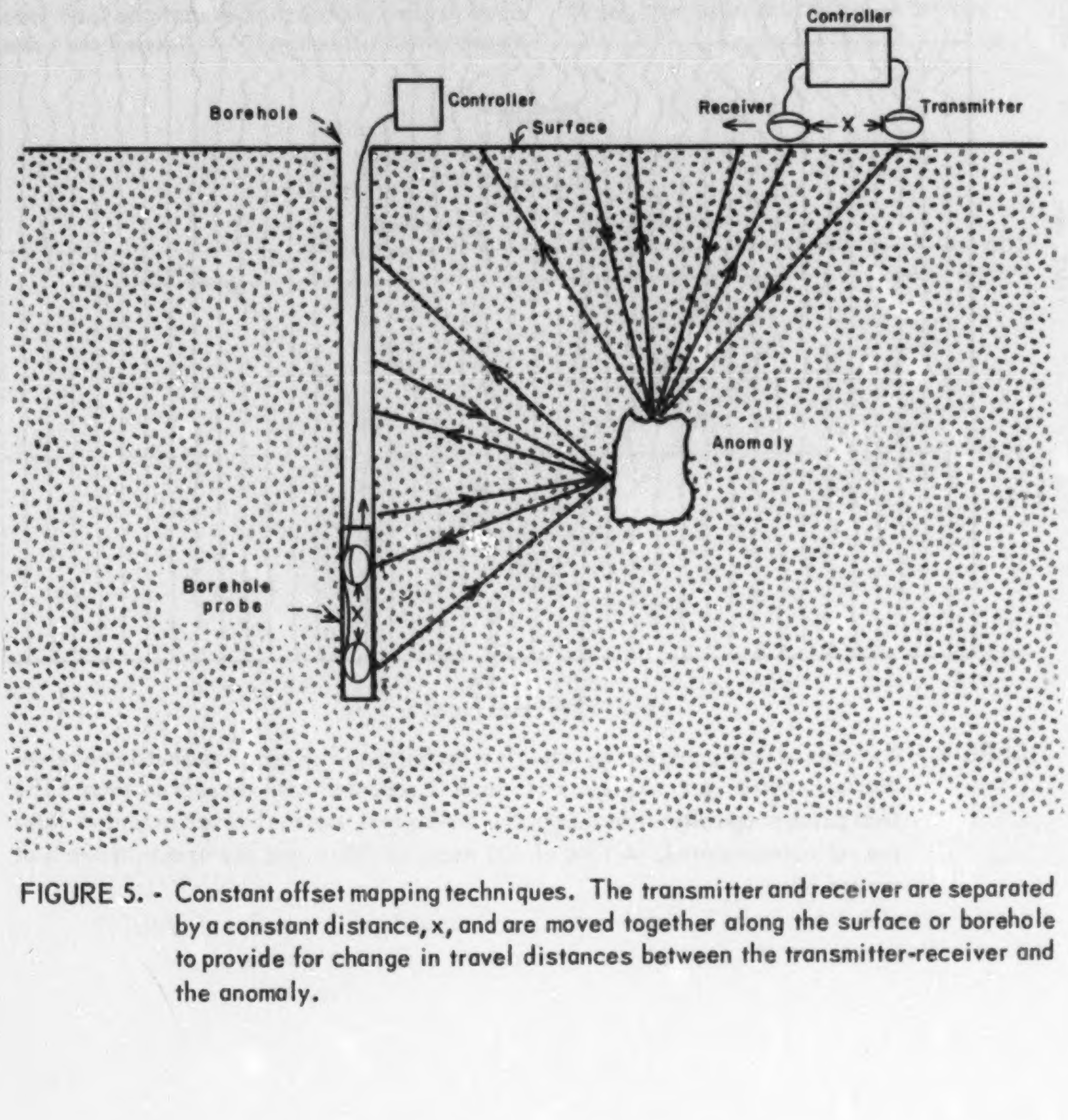
Controller
Receiver
X
Transmitter
Controller
Borehole
Surface
Anomaly
Borehole probe
X

FIGURE 5. - Constant offset mapping techniques. The transmitter and receiver are separated by a constant distance, x, and are moved together along the surface or borehole to provide for change in travel distances between the transmitter-receiver and the anomaly.

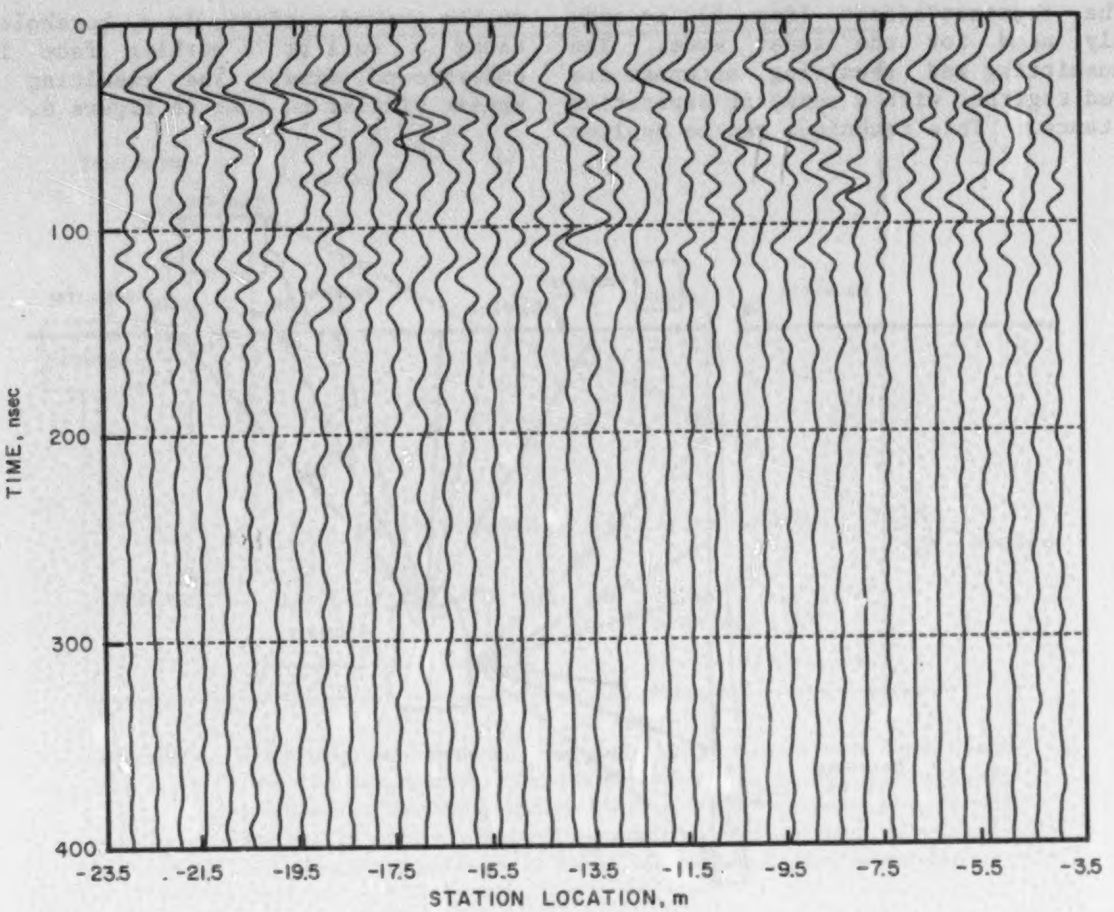

FIGURE 6. - Example of data using a constant offset technique. For each transmitter-receiver station location, a short pulse wave form is recorded with respect to time. The first pulse shape after time zero is the air-surface wave. Pulses recorded later are reflection related. A time of 400 nsec, in this case, represents a depth of around 30 m.

The common depth point (CDP) mapping technique is shown in figure 7. This technique maps an anomaly by separating the transmitting antenna from the receiving antenna. The target or anomaly remains fixed, but the travel distance between the antennas increases with each antenna movement. Figure 8 illustrates the CDP approach as it was applied in a coal mine using the synthetic-pulse method. Using the change in arrival times for the reflections, and knowing or estimating the velocity, it is possible to determine the distance of the anomaly from the transmitting network.

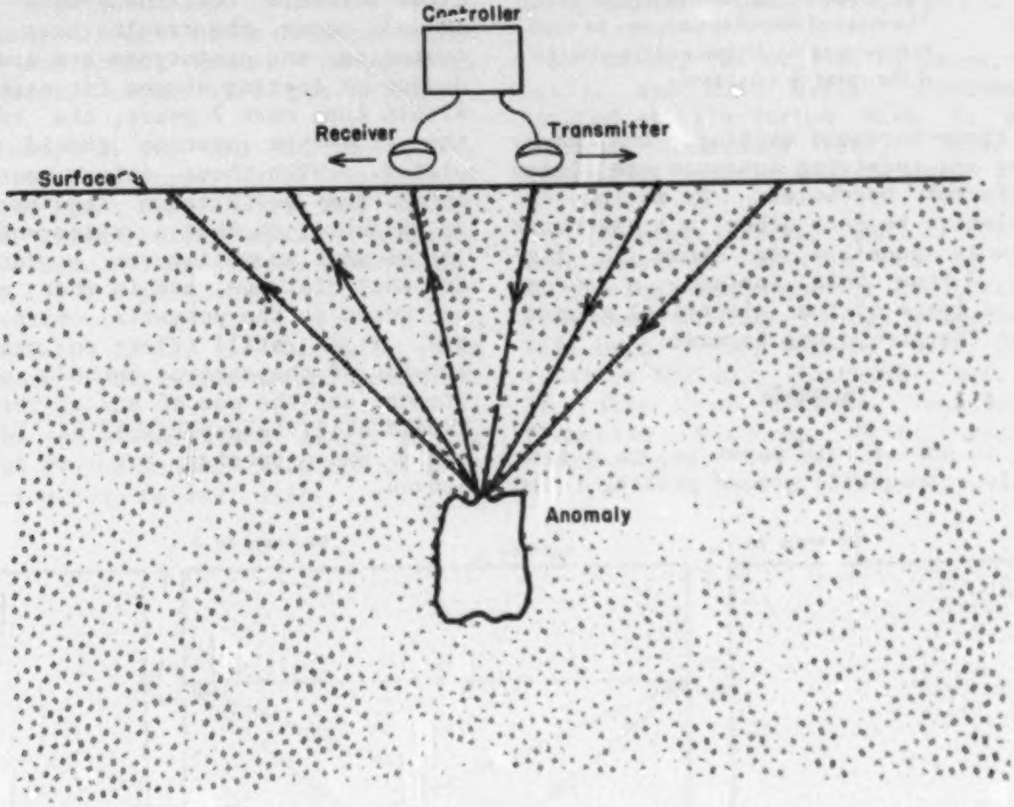

FIGURE 7. - Common depth point mapping technique. The transmitter and receiver are moved apart from each other, causing a change in travel paths for each reading.

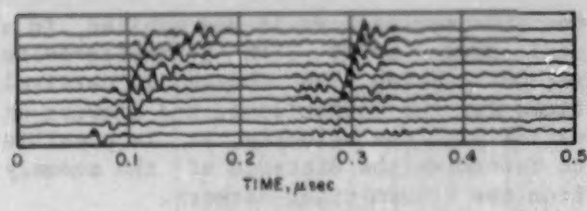

FIGURE 8. - Reflection in coal using common depth mapping. Pulses occurring around 3 μsec are reflection through 50 ft of coal. The slope of the reflections can be used to determine the distance of the reflector if the velocity is known.

For cross-borehole mapping, the transmitting and receiving antennas are placed in different boreholes. By moving the two relative to each other, a cross-hatch pattern is possible, as shown in figure 9. From this pattern, a reconstructed image of the electrical properties of the ground can be made.

SUMMARY

The intent of this paper is to introduce electromagnetic ground probing radar methods to those unfamiliar with the technique. This was done by showing some of the different methods and how they are applied in detecting and mapping geological features and potential hazards. The Bureau has now completed feasibility tests of short-pulse surface and borehole techniques, synthetic-pulse methods for detecting hazards in coal mines, and cross-borehole continuous-wave studies. In all cases, the results have been encouraging, and prototypes are now in the design or testing stages for each method. Within the next 2 years, the results of the prototype systems should be completed. With these future tests, it is hoped that performance figures can be established, so that a better understanding of what anomalies can be determined, at what distance, and in what media can be known to the potential users. As of now, it is still tricky to predict the results of the various methods and techniques, and the use of any of the methods should still be approached as being possible and promising, but not completely proven.

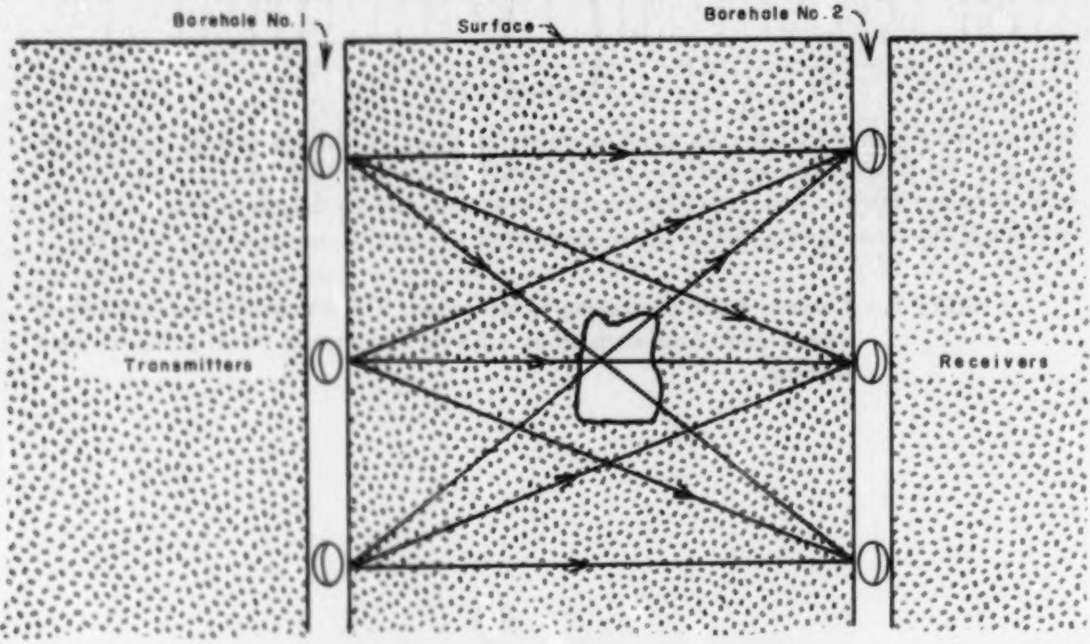

FIGURE 9. - Cross borehole mapping technique. By moving the transmitter and receiver to various locations, a variety of travel paths are possible. By comparing the different received signals, detailed analysis of the material between holes is possible.

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[2] Items 1, 4, and 7 are available for reference at the Denver Research Center, Bureau of Mines, Denver, Colo.

IN SITU NEUTRON ACTIVATION ANALYSIS

By George J. Schneider[1]

ABSTRACT

An in situ neutron activation analysis system has been developed and tested in several mineral deposits. The 2-in-diam borehole logging sonde in the system contains a microprocessor-controlled, high-resolution, gamma-ray spectrometer with digital data transmission to the surface on 4-H-0 logging cable. Californium is used as a neutron source for activation. The gamma-ray spectrometer has been adapted to commercial service for assaying uranium ores in disequilibrium.

INTRODUCTION

In September 1976, after evaluating competitive proposals, the Bureau of Mines let a contract[2] to Princeton Gamma-Tech, Inc., to design a unique borehole assaying system. Their research has resulted in the development and commercial availability of a logging system that still defines the state-of-the-art for in-place ore grade analysis. The logging sonde includes a high-resolution intrinsic germanium detector cooled by a solid or melting cryogen; a 4,000-channel, microprocessor-controlled, multichannel analyzer; and a digital communications link along industry standard 4-H-0 logging cable to a microcomputer-based data storage and display system at the surface. Small quantities of the man-made isotope, californium-252, are used as a source of neutrons for activation.

The advantages of downhole data processing are demonstrated by the measured 2 keV resolution at 1.33 MeV, of the borehole gamma-ray spectrometer over 3,000 ft of 4-H-0 logging cable. Previous systems have only provided about 14 keV resolution over 1,000 ft of coaxial cable at the same energy. The practical benefit is virtually laboratory-quality gamma-ray spectra for in situ analysis. Figure 1 is a complete prompt capture gamma-ray spectrum from a magnetite deposit acquired by the sonde with a 2.2-μg californium-252 source in a 5-in-diam borehole; it illustrates the high-resolution spectra routinely obtained.

The reliability of the system is demonstrated by the daily commercial use of derivative systems for assaying uranium ores. The contractor has operated several systems in regular service for nearly 2 years. A derivative of the multichannel analyzer design has also been adapted to a commercial laboratory analyzer.

NEUTRON ACTIVATION ANALYSIS

Neutron activation analysis is an inherently effective technique for borehole assaying. The method is, in general, absolute and volumetric, and it exceeds mining industry requirements for elemental sensitivity in near-real-time analysis. A source of neutrons, californium-252 in the present Bureau system, irradiates the rock surrounding an access borehole. The neutrons are slowed to thermal equilibrium with the rock mass from an initial energy between 1 MeV and 8 MeV by many elastic and inelastic collisions with the nuclei of individual atoms in the rock. Elastic scattering describes a collision of a fast neutron and an atomic nucleus, where kinetic energy is conserved. In inelastic scattering, a net loss of kinetic energy occurs. An inelastic collision raises the energy of the atomic nucleus to an

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[1] Geologist, Denver Research Center, Bureau of Mines, Denver, Colo.

[2] Contract H0262045, "A Borehole Probe for In Situ Neutron Activation Analysis."

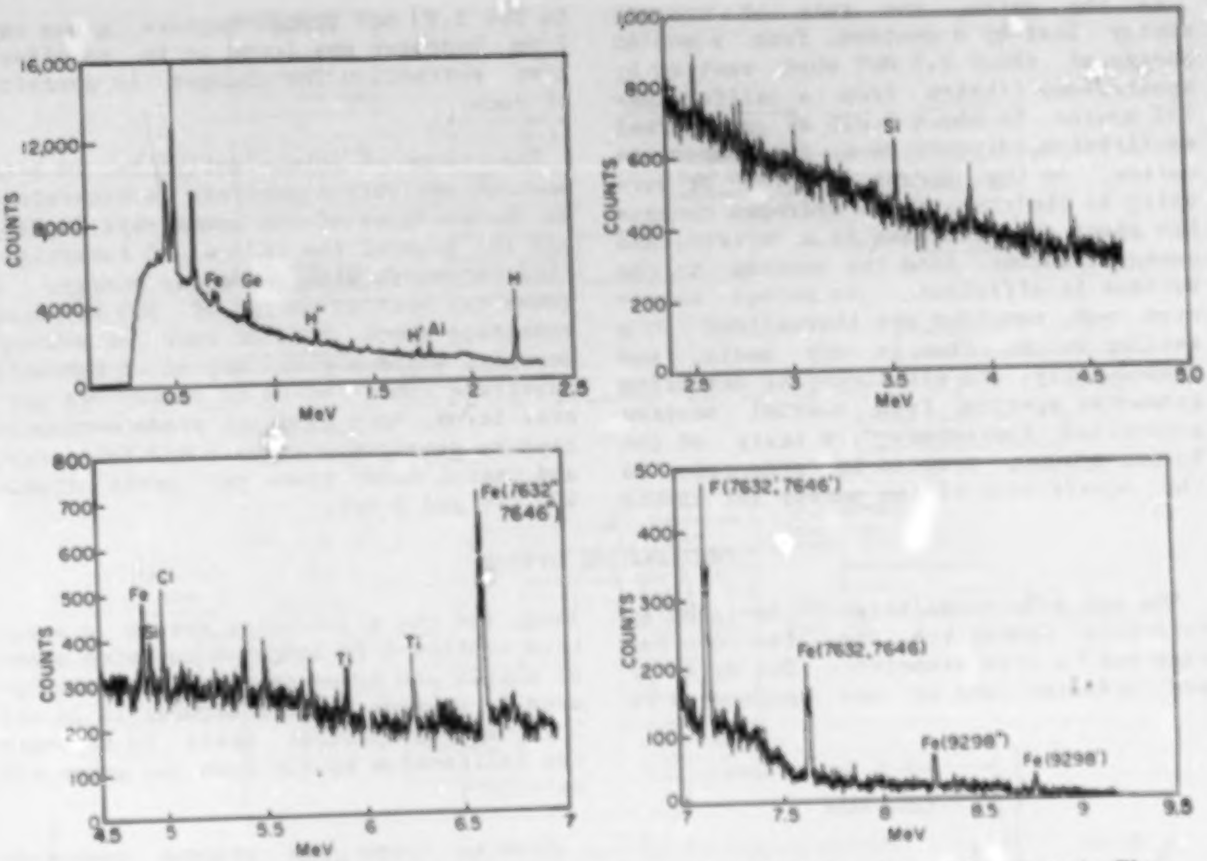

FIGURE 1. - In situ gamma ray spectra from magnetite (iron ore) deposit near Dover, N.J. This is a complete 4,000-channel prompt-capture gamma ray spectrum obtained with the logging system (in four parts).

unstable state and the nucleus returns to a stable state through the emission of gamma rays at discrete energies.

After losing energy in many collisions, a neutron reaches thermal equilibrium with the rock mass. Thermal neutrons are captured by atomic nuclei in the rock according to well defined probability or cross section for each isotope. Neutron capture raises the energy of a nucleus to an unstable state. With only a few exceptions all the naturally occurring isotopes return to a stable state by the emission of one or more gamma rays at discrete energies; and in decay reactions, by the emission of another subatomic particle. An individual nucleus can return to a stable state by only one, or one sequence of, several available reaction paths, but a group of nuclei of the same isotope will follow all available reaction paths according to very precisely known ratios.

Prompt-capture gamma rays are produced by the immediate release of energy after thermal neutron capture. Decay gamma rays are produced by reactions with a specific statistical half life. Thermal neutron activation, both prompt capture and decay, is in practical terms the most important class of neutron activation reactions for in situ analysis, because these reactions have the highest cross section or probability of occurrence for most isotopes and are the most prolific producers of gamma rays.

In the earth, the rate of kinetic energy loss by a neutron, from a median energy of about 2.5 MeV when emitted by spontaneous fission from a californium-252 source to about 0.025 eV at thermal equilibrium, depends to a first approximation, on the amount of water or porosity in the rock. The hydrogen nucleus has about the same mass as a neutron, and energy transfer from the neutron to the nucleus is efficient. In porous saturated rock, neutrons are thermalized in a smaller volume than in dry media, and consequently, the efficiency of detecting gamma-ray spectra from thermal neutron activation increases. In tests of the Bureau system, a ratio in proportion to the square root of the number of counts in the 2.23 MeV prompt-capture gamma ray from hydrogen was found to be an effective correction for changes in porosity of rock.

The volume of investigation for in situ neutron activation analysis is determined by the energies of the gamma rays logged, and in general the radius of investigation increases with gamma-ray energy. A gamma ray with an energy of 500 keV will penetrate about 5 cm of rock of average density, while a gamma ray of 5 MeV will penetrate about 40 cm of rock. In general terms, most elements produce prompt-capture gamma rays above 3 MeV in energy, and useful decay gamma ray peaks between 400 keV and 3 MeV.

THE LOGGING SYSTEM

The borehole sonde (fig. 2) designed by Princeton Gamma-Tech, Inc., for the Bureau has a 2-in diameter. The detector and cryostat are in one section 4 ft long, and the electronics are in a separate section 6 ft long. A tungsten shadow shield and nylon spacers were configured in various lengths between 18 in and 5 ft during specific tests to separate the californium source from the gamma-ray detector.

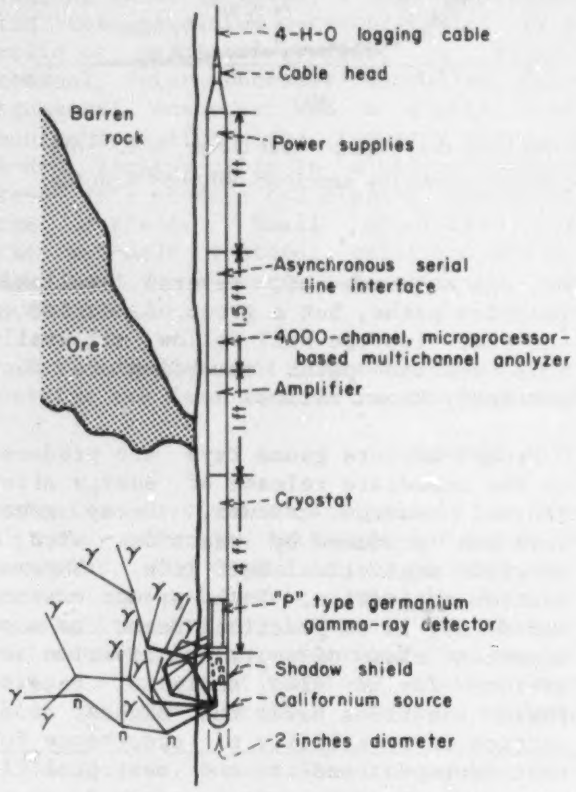

FIGURE 2. - The logging sonde.

Several Freon and propane compounds frozen by liquid nitrogen and, during one test, a solid slug of copper cooled to 77 K have been used to cool the intrinsic germanium detector. The cryogens are contained in removable canisters that can be inserted and removed from the sonde in a few minutes. A single canister of Freon-22 permits over 8 hr of logging.

The electronics package in the sonde contains a preamplifier, an amplifier, a high-voltage supply for detector bias, a successive approximation type analog-to-digital (ADC) converter, low-voltage power supplies for the components, a Motorola 6800 microprocessor with 4,096 eight-bit words of memory, and a half duplex serial cable transceiver. A total of 3,968 words of memory were used for data storage. Conversion time, independent of pulse height, was 16.6 msec for the ADC.

The operation of the system is shown schematically in figure 3. The intrinsic

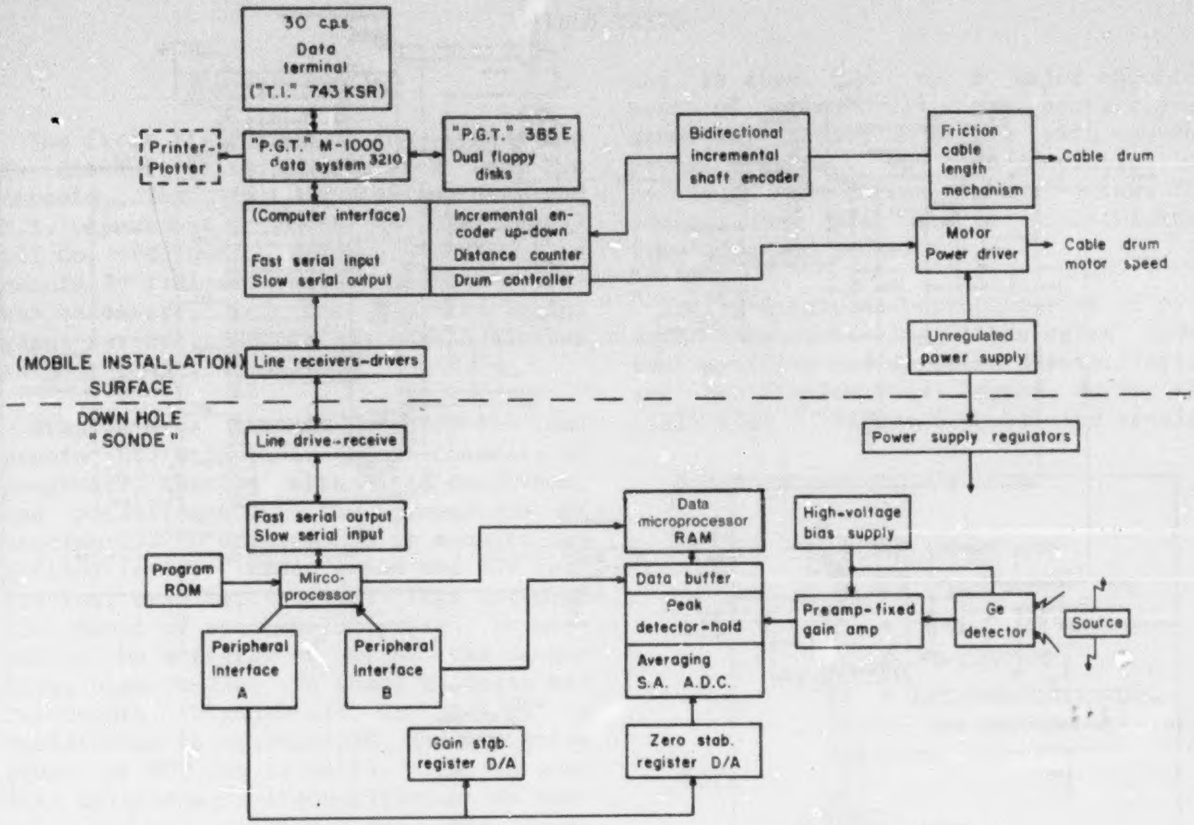
30 cps
Data
terminal
("T.I." 743 KSR)
Printer-
Plotter
"P.G.T." M-1000
data system 3210
"P.G.T." 385 E
Dual floppy
disks
Bidirectional
incremental
shaft encoder
Friction
cable
length
mechanism
Cable drum
(Computer interface)
Fast serial input
Slow serial output
Incremental encoder up-down
Distance counter
Drum controller
Motor
Power driver
Cable drum
motor speed
Unregulated
power supply
(MOBILE INSTALLATION)
SURFACE
Line receivers-drivers
DOWN HOLE
"SONDE"
Line drive-receive
Power supply regulators
Fast serial output
Slow serial input
Data
microprocessor
RAM
High-voltage
bias supply
Program
ROM
Mirco-
processor
Data buffer
Peak
detector-hold
Averaging
S.A. A.D.C.
Preamp-fixed
gain amp
Ge
detector
Source
Peripheral
interface
A
Peripheral
interface
B
Gain stab.
register D/A
Zero stab.
register D/A

FIGURE 3. - Operating logic for the logging system.

germanium detector converts the energy of a gamma ray within the detector to a voltage pulse proportional in amplitude to the energy of the incident gamma ray. The voltage signal is filtered, shaped, and amplified before input to the ADC. A digital number equivalent to the amplitude of the voltage pulse is transmitted from the ADC to the microprocessor. The microprocessor stores the number from the ADC as a single count in an appropriate location in memory corresponding to the energy of the incident gamma ray. Data are accumulated and stored in memory for several seconds at an effective rate of 10,000 to 20,000 counts per second, and then transmitted along 4-H-0 logging cable to a microcomputer at the surface. The microprocessor transmits the number of counts in each channel in succession and resets each channel back to zero. Transmission is interrupted by any new event from the ADC and resumes after the incoming event is processed and stored in memory. Data transmission is over a single wire of four-conductor 4-H-0 cable at a rate of 31,800 bits per second (31.8 baud).

A microcomputer at the surface receives the data at a serial line interface off the logging cable. The data are stored on magnetic tape or disks, and processed for display on a terminal in the logging van whenever convenient for the operator. The microcomputer at the surface controls the cable which through a digital-to-analog converter (DAC). The circuitry and components for the microprocessor based multichannel analyzer, preamplifier, amplifier, and power supplies in the sonde are available in the previously released Phase I project report at Bureau libraries.

Revised circuit diagrams for the ADC are shown in figure 4, and for the microprocessor-based analyzer in figure 5.

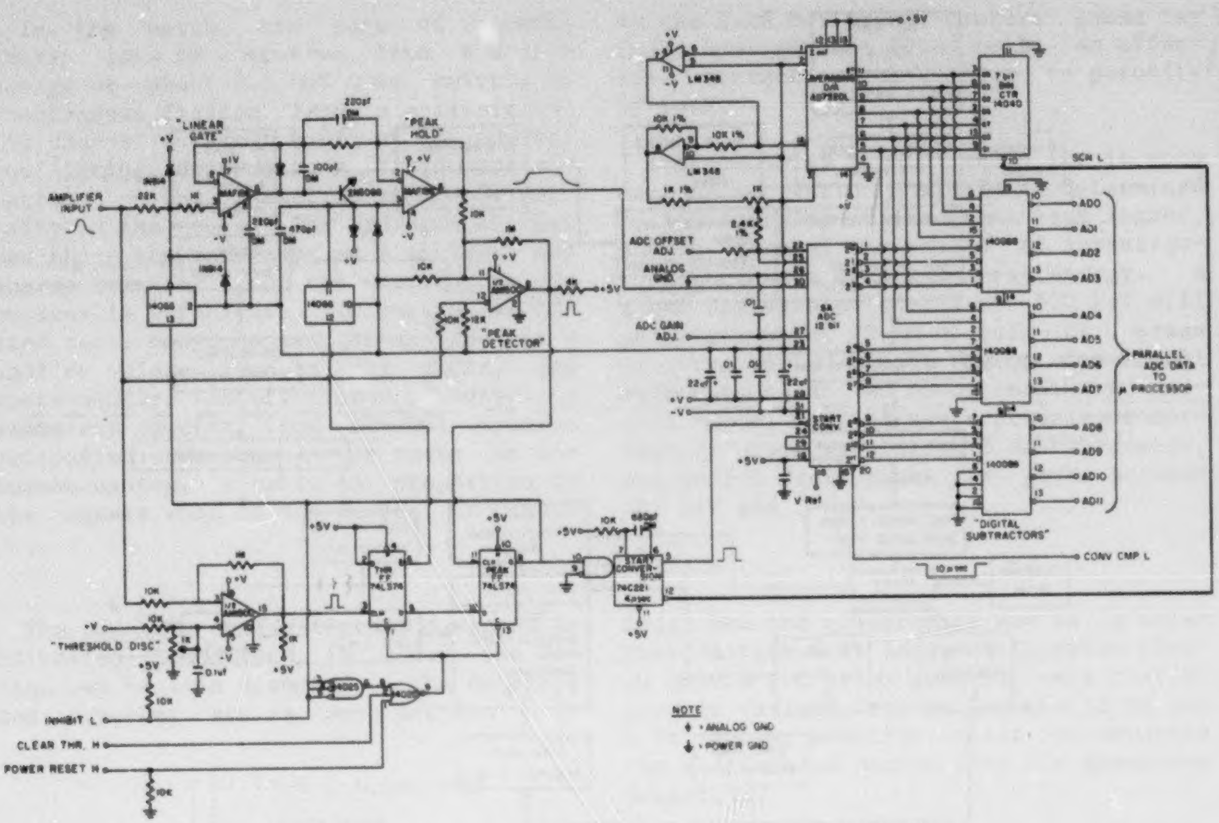
AMPLIFIER INPUT
"LINEAR GATE"
"PEAK HOLD"
"PEAK DETECTOR"
"THRESHOLD DISC"
INHIBIT L
CLEAR THR. H
POWER RESET H
LM348
D/A AD7520L
ADC OFFSET ADJ.
ANALOG GND.
ADC GAIN ADJ.
SA ADC 12 bit
START CONV.
74C221
THR FF 74LS74
CLR PEAK FF 74LS74
START CONVER-SION
7 bit Bin CTR 14040
SCH L
14008
AD0
AD1
AD2
AD3
AD4
AD5
AD6
AD7
AD8
AD9
AD10
AD11
PARALLEL ADC DATA TO µ PROCESSOR
"DIGITAL SUBTRACTORS"
CONV CMP L
V Ref
NOTE
= ANALOG GND
= POWER GND

FIGURE 4. - Circuitry and components for the downhole ADC.

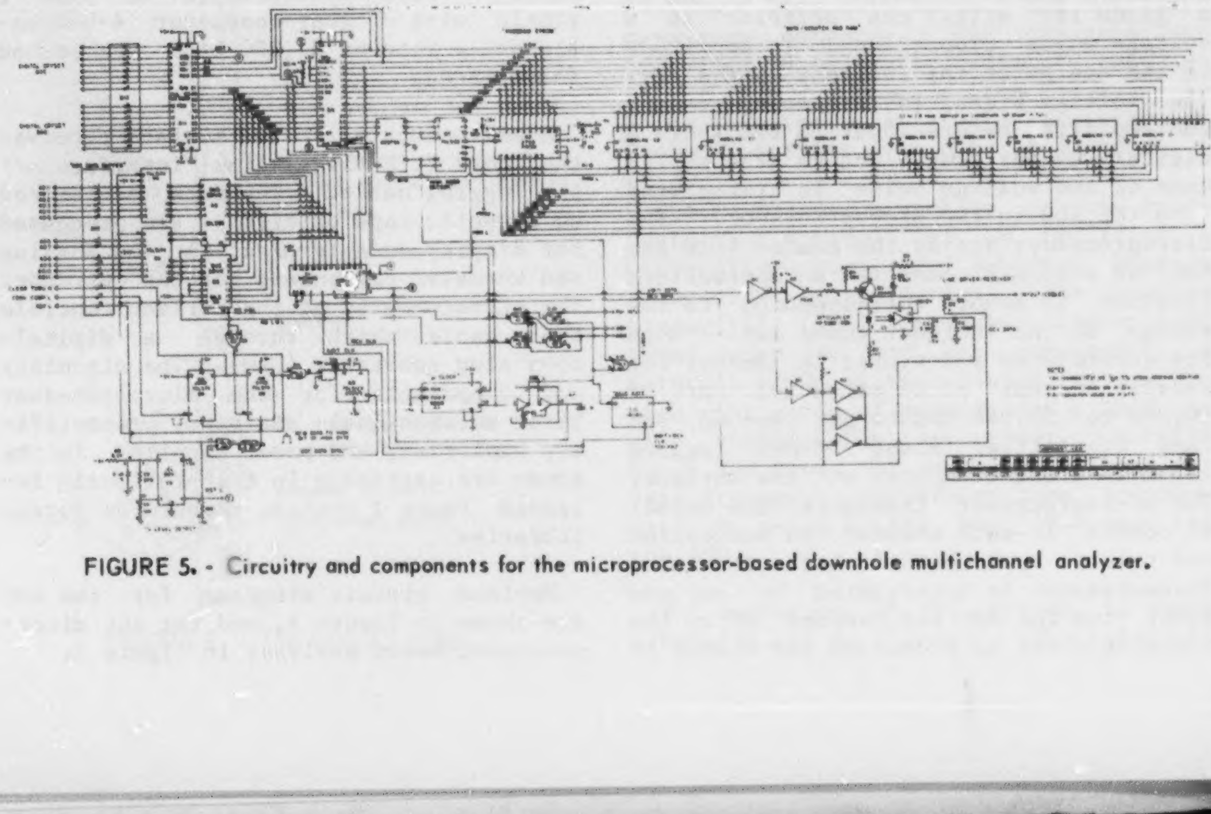

FIGURE 5. - Circuitry and components for the microprocessor-based downhole multichannel analyzer.

FIELD TESTS

Uranium

The first field test of the system was in disequilibrium uranium ores near San Antonio, Tex., in cooperation with the U.S. Department of Energy and Continental Oil Co. (CONOCO). Because uranium is naturally radioactive, no neutron source was necessary, and the high-resolution gamma-ray spectrometer could be evaluated without complication.

Uranium ores contain uranium-238 and uranium-235 with their decay products or daughters, thorium with its daughters, and potassium-40. The proportion of uranium-235 to uranium-238 is accepted as a fixed ratio. Gross gamma and KUT (potassium, uranium, thorium) logs estimate the amount of uranium-238 present by measuring the activity of one of the daughters, bismuth-214. In ideal geologic environments, bismuth-214 is present in equilibrium to uranium-238, and the gross gamma or KUT log is valid. But in many real environments disequilibrium is common between uranium and the long-half-life daughters because of differential leaching or other mechanisms for removal of uranium or its daughters selectively. The measurement of equivalent uranium by conventional methods in these environments is in error since the concentration of uranium and bismuth-214 are not in proportion. In south Texas and parts of New Mexico and Wyoming, disequilibrium ores are typical, requiring expensive core drilling for economic evaluation of the deposits and mine planning.

Two daughters of uranium-238 establish equilibrium with the parent isotope in a few months and are therefore always in equilibrium under geologic conditions. The first daughter of uranium-238, thorium-234, has a half life of 24.1 days and decays to protactinium-234. Protactinium-234 decays with a 1.17-min half-life, accompanied by the emission of a gamma ray at 1.001 MeV. Because the gamma-ray is produced only 0.59 pct of the time an atom of uranium-238 decays, and is also close to a major spectral peak of bismuth-214, the protactinium gamma ray cannot be logged with conventional systems. The high resolution inherent in the Bureau system allows the protactinium peak to be discriminated from adjacent peaks.

Thirty boreholes representative of several uranium-bearing lithologies under both positive and negative disequilibrium and equilibrium were logged during the field test. Figure 6 shows the results

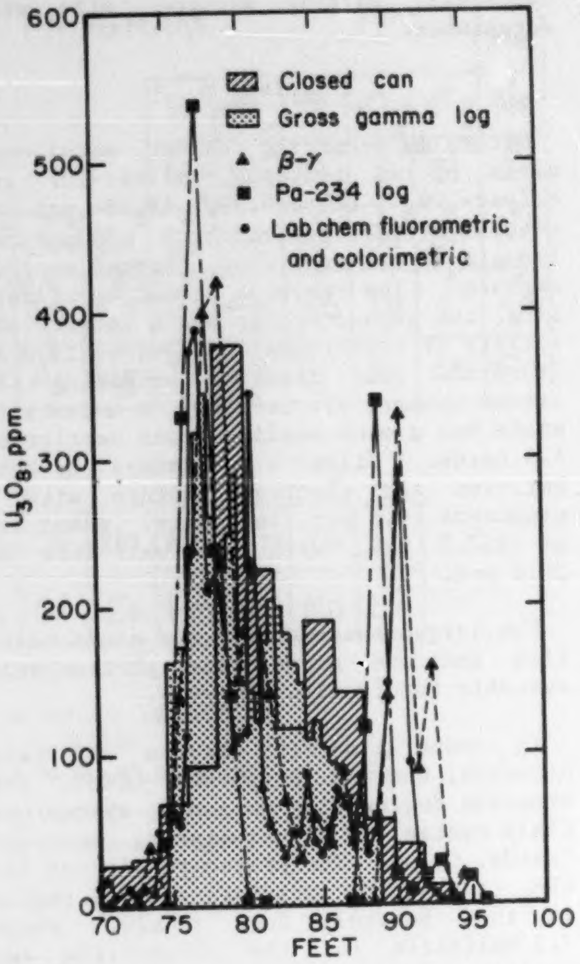

FIGURE 6. - Laboratory and in situ analysis in Texas uranium ore. The Pa-234 (protactinium) log provides a correct in situ assay through an ore zone where uranium is in disequilibrium with the long-half-life daughters.

of gross gamma logging confirmed by closed can laboratory analysis of core sample, and the protactinium (Pa-234) log confirmed by beta minus gamma (β-γ), colorimetric, and fluoriometric laboratory assays of core samples from a test borehole in Texas uranium ore. Note the top of the ore zone is in equilibrium. Uranium has been leached from the center of the ore zone and redeposited immediately below the leached zone. The Pa-234 log shows the distribution of uranium correctly, while the gross gamma log does not. The tests conclusively demonstrated the feasibility of the technique. The contractor now offers the method as a commercial logging service with wide acceptance.

Silver

Naturally occurring silver metal consists of two isotopes, silver-107 and silver-109. Silver-109, 48.18 pct of natural silver, activates to produce two isomers of silver-110 by thermal neutron capture. The reaction producing silver-110g, the ground state, has a large probability of occurrence or cross-section of 88 barns. The reaction producing the second isomer, silver-110m, a metastable state has a much smaller cross section of 4.4 barns. Silver-110g decays by beta emission and electron capture with a prominent (4.5 pct intensity) gamma ray at 657.7 keV with a half-life of 24.4 sec.

The large cross-section and short half-life indicate a reaction particularly suitable for in situ analysis.

In cooperation with Chevron Oil Field Research, Chevron Resources (CRC), and Minerals Engineering Co., the system was field tested in silver bearing ore near Creede, Colo. The procedure adopted for the test was to irradiate a 1-ft region in the borehole for 1 min, about 2.5 half-life for the silver-110g reaction, and then to move the germanium detector into the region for 1-min to measure the activity of silver-110g. The gamma ray spectrum was calibrated over the region including the 657.7 keV silver-110g gamma ray by fixing the location of the 511-keV positron annihilation peak and the 1,778-keV decay peak from aluminum-28. The 2,223-keV prompt capture gamma ray from hydrogen was used to normalize the data for formation porosity. The foot-by-foot data were averaged into assays representative of 5 ft-long intervals to compare with laboratory assays on samples from the drill holes. The good agreement between the in situ analysis and the laboratory assays is indicated in figure 7, the results from logging one of the test boreholes.

Other Field Tests

Field tests were also conducted in iron ores in cooperation with Halecrest Mining Co., copper porphyry ore in cooperation with Kennecott Corp., metallurgical coal in cooperation with the United States Steel Company and the U.S. Geological Survey, and gold ore in cooperation with Golden Cycle Mining Co., Texasgulf Corp., and the U.S. Geological Survey. The results of these tests will be discussed in detail in future publications.

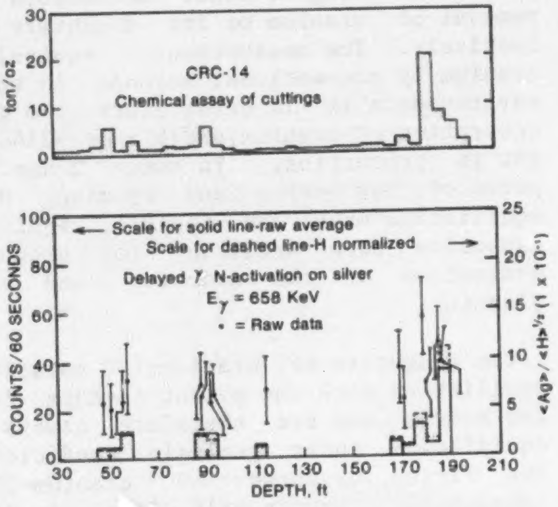

FIGURE 7. - Compilation of 5-ft average silver assay values from foot-by-foot measurements. The effect of the hydrogen correction for porosity is also shown.

FUTURE RESEARCH

Further development of borehole neutron activation analysis is planned by replacing the californium sources in the system described with a pulsed, sealed-tube neutron generator, and a small linear accelerator. The use of an accelerator as a neutron source eliminated the shielding required to protect operating personnel, since neutrons are produced only when potential, typically 100 keV, is applied to the target in the generator. Sensitivity will improve for decay gamma-ray logging because of increased neutron flux, and for prompt-capture gamma-ray logging because the spectra are collected between pulses in much improved background. Also, inelastic scattering reactions will be usable because of the neutron energy available with an accelerator type source. Figures 8 and 9 show the estimated sensitivity obtainable for most elements of economic interest with a fully developed system by in situ neutron activation analysis and other related in situ assaying methods within real operational constraints.

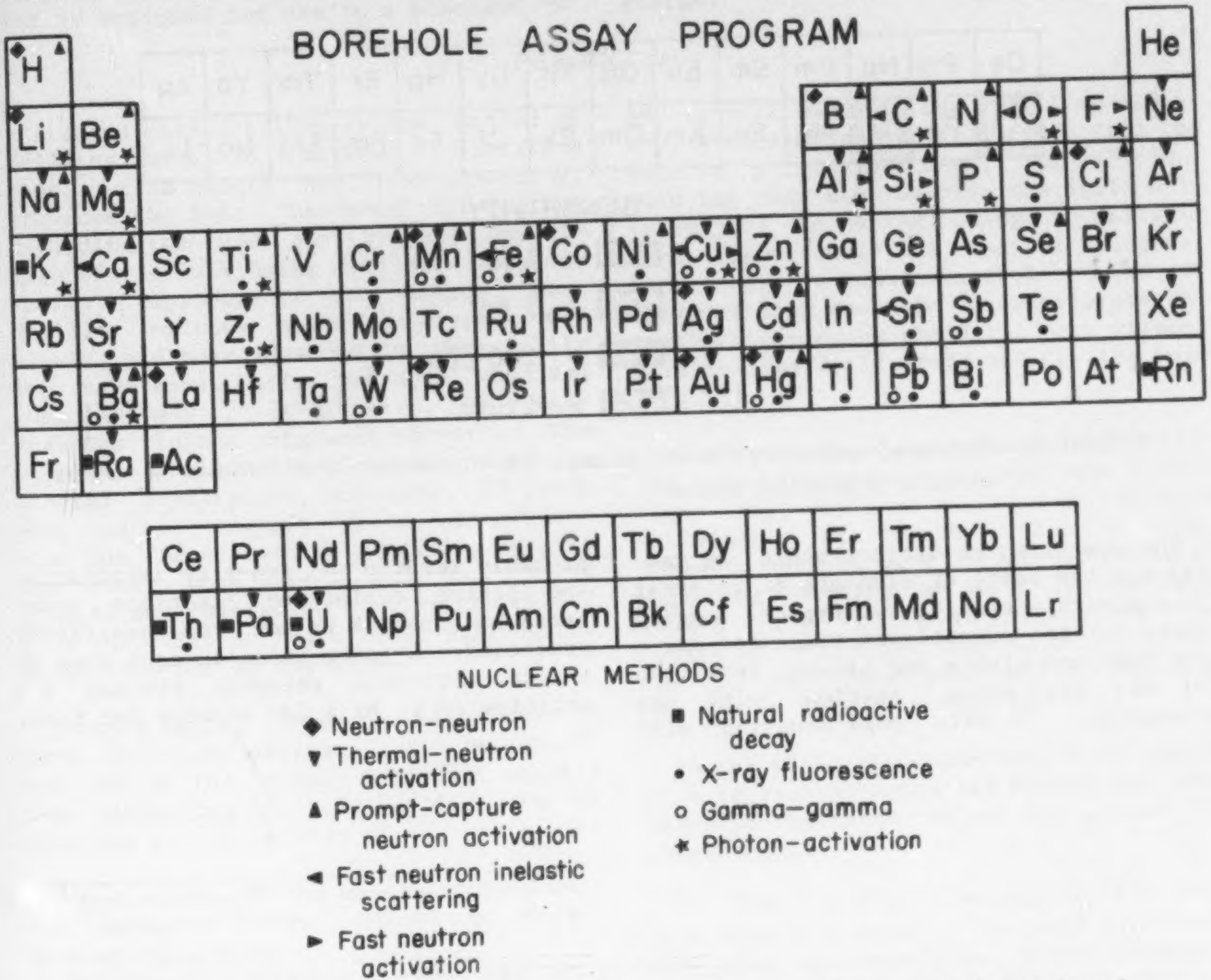
BOREHOLE ASSAY PROGRAM
H He
Li Be B C N O F Ne
Na Mg Al Si P S Cl Ar
K Ca Sc Ti V Cr Mn Fe Co Ni Cu Zn Ga Ge As Se Br Kr
Rb Sr Y Zr Nb Mo Tc Ru Rh Pd Ag Cd In Sn Sb Te I Xe
Cs Ba La Hf Ta W Re Os Ir Pt Au Hg Tl Pb Bi Po At Rn
Fr Ra Ac
Ce Pr Nd Pm Sm Eu Gd Tb Dy Ho Er Tm Yb Lu
Th Pa U Np Pu Am Cm Bk Cf Es Fm Md No Lr
NUCLEAR METHODS
◆ Neutron-neutron
▼ Thermal-neutron activation
▲ Prompt-capture neutron activation
◀ Fast neutron inelastic scattering
▶ Fast neutron activation
■ Natural radioactive decay
● X-ray fluorescence
○ Gamma-gamma
★ Photon-activation

FIGURE 8. - In situ assaying methods for elements of economic interest. Several methods have been demonstrated as applicable. Selection of a method depends on whether the assay is to be done in boreholes, at the working face, or in haulage and on the sensitivity required. The neutron-based methods are in the left column.

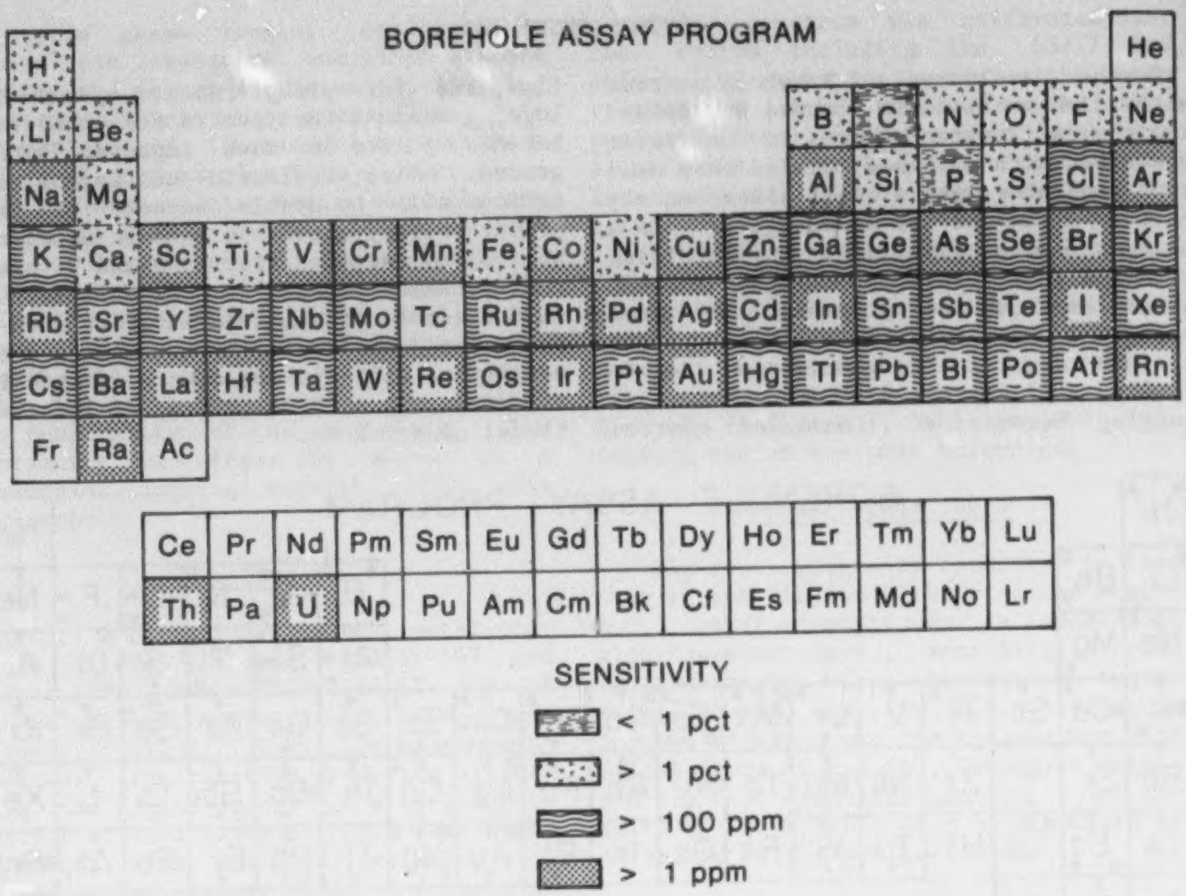

FIGURE 9. - Estimated sensitivity that can be expected with neutron-based methods for various elements of economic interest.

The operating conditions would be varied for the suite of elements of interest in a mineral deposit, but typical logging rates for the sensitivity indicated are a few feet per minute and always less than 10 min per assay station with one exception. In situ gold assaying will probably require a two-step irradiation and spectra collection technique separated by about 8 hr for the sensitivity predicted. Logging rates in each step of the in situ gold assaying process are anticipated to be a few minutes per foot.

DEVELOPMENT OF AN IN-HOLE REPLACEABLE DIAMOND CORE BIT SYSTEM

By W. C. Larson,[1] W. W. Svendsen,[2] R. E. Cozad,[3] and J. R. Hoffmeister[4]

ABSTRACT

A one-piece diamond core drill bit has been developed that can be removed for inspection or replacement without pulling the drill rods from the hole. The replaceable bit system consists of two basic subsystems: The removable core bit, and the down-hole equipment necessary to replace the bit. The replaceable bit is designed for use in a standard "N" size wireline core drilling system. The system has undergone laboratory and full-scale field testing and is currently undergoing additional long-term field testing and engineering evaluation by the Longyear Co., Minneapolis, Minn. This report describes the replaceable bit system and summarizes its current status.

INTRODUCTION

Diamond core drilling is one method of drilling used in mineral exploration that, while costly and time consuming, provides the best "hands-on" information regarding the type, quality, and composition of the rock being drilled. Historically, improvements in diamond core drilling methods have developed very slowly, and most advances in core drilling have come, after years of need and long periods of development, as breakthroughs in the state-of-the-art. The wireline core drilling system used in mineral exploration, now about 25 years old, is one example where significant time has been saved in drilling because the inner core barrel can be removed (raised and lowered on a wireline cable) from the drill string without pulling the drill rods out of the hole.

Another major, time-consuming task in core drilling, estimated at between 5% and 10% of the total available working time (depending on hole depth), is the changing or inspection of the core bit. As is well known in the drilling industry, withdrawing the drill rods from the hole is a laborious task. In addition, once the rods are removed, the drill hole is subject to caving owing to unstable rock formations. A logical advance, therefore, in the state-of-the-art of core drilling would be to incorporate the idea of replacing the drill bit without pulling the drill rods out of the hole (fig. 1).

Many research organizations and drilling companies have recognized the advantages that could be realized by using a replaceable bit system. For example, domestic patents on a variety of retractable bit concepts go back to the late 1800's and illustrate the long-time desire to reduce the nonproductive drilling time associated with pulling the rods out of the hole to change the bit.

In the past, the commercial development of a retractable core bit system has been hampered by three broad categories of shortcomings:

1. The cutting elements of the bits that were developed in the past retracted each time the inner core barrel assembly was pulled whether the bit was worn or not. With this type of system an extremely high reliability factor is necessary to achieve an efficient operation.

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[2] Technical director, Longyear Co., Minneapolis, Minn.

[3] Project leader, Doerfer, Cedar Falls, Iowa.

[4] Consultant, Minneapolis, Minn.

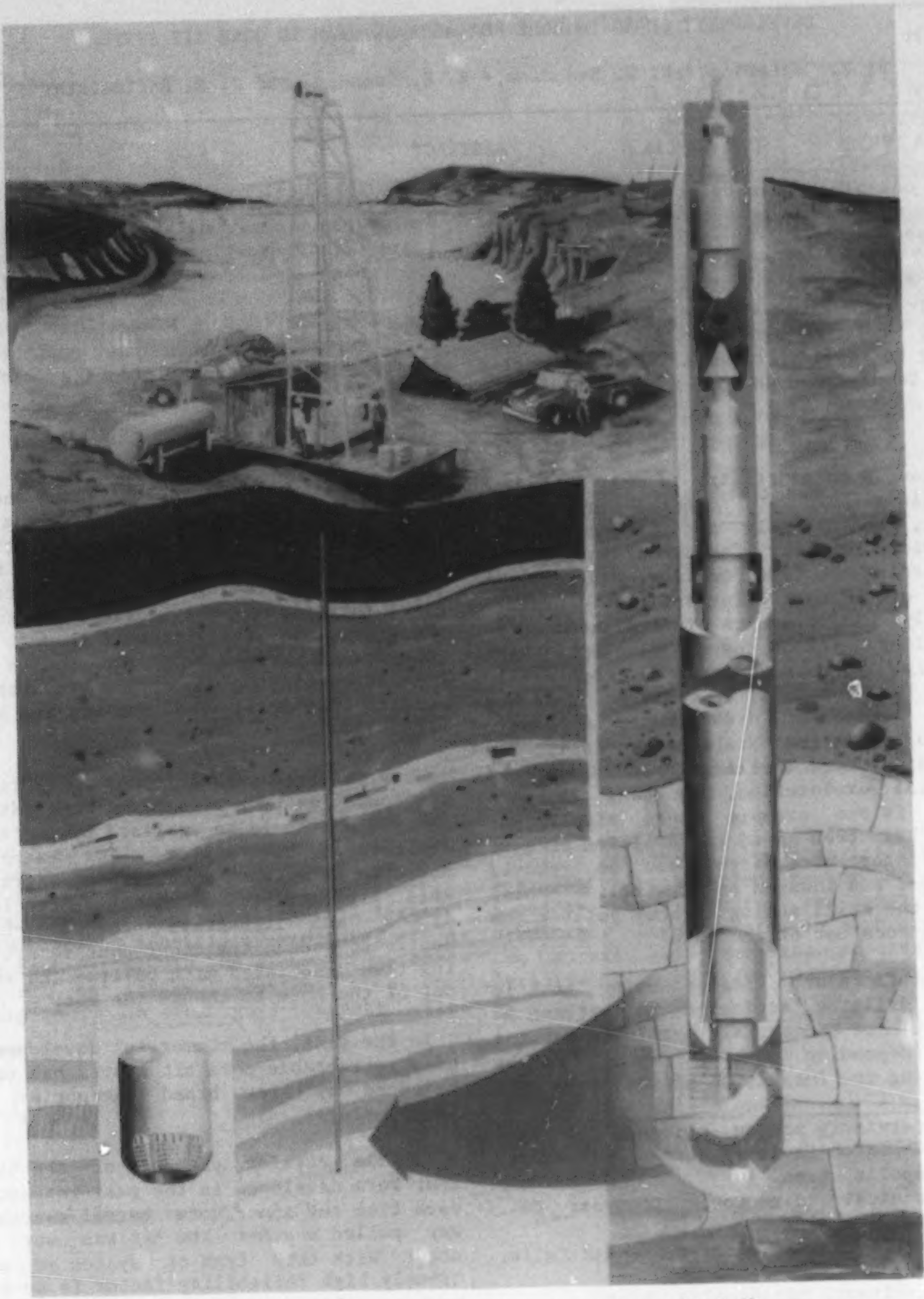


FIGURE 1. - Schematic cross section of the retractable core bit drilling system.

2. The core bit, a consumable component, was complicated, leading to relatively expensive machining, close tolerances, and inherent operating difficulties.

3. The bit and associated retraction mechanisms were not rugged enough to endure the rigors of normal drilling operations.

In order to increase core drilling productivity and advance the state-of-the-art in core drilling technology, the Bureau of Mines, through its contract research program and in-house capabilities, designed a down-hole replaceable core bit drilling system. The system has been fabricated and field tested.

GENERAL DESCRIPTION

This system functions as a conventional wireline core drilling system in all aspects until it becomes necessary or desirable to change the bit. After the inner core barrel is removed from the drill rods, the bit retraction tool is lowered through the rods by means of the wireline cable until it seats in the landing rig of the outer tube assembly. At the surface, a packer is applied to the top of the drill rods and fluid pressure is applied utilizing the available pumping system. This procedure locks the tool in place for the retraction operation and assures that the tool is in the proper orientation. Raising the wireline cable, which remains attached to the tool, allows the retraction mechanism to lock into the bit, push it clear of the bit holder, and rotate it into position for passing through the drill rods (fig. 2) to the surface. The next step is to inspect and/or replace the bit and reverse the procedure as described above using the bit insertion tool.

Both retraction-insertion tools are alike in appearance, size, and internal construction. The only difference is in the drive mechanism, which transforms the upward forces of the wireline cable into the combination of forces and movements needed to remove or replace the bit. In effect the internal drive mechanisms of the tools are reversed.

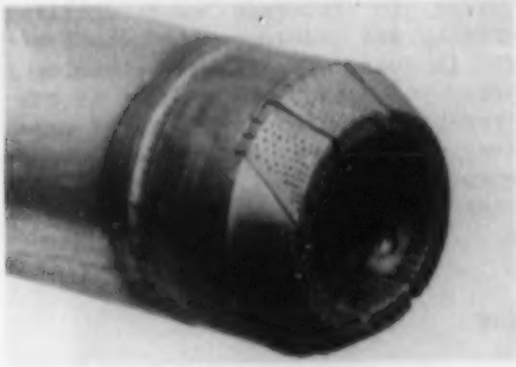
System Components and Operating Procedures

The bit is a one-piece diamond set element position for drilling by two steel lugs and held by a locking device incorporated within the bit holder of the outer tube assembly of the wireline core barrel. Its external configuration is similar to that of a wireline core bit except for omission of two portions that provide the interface with the driving lugs (fig. 3).

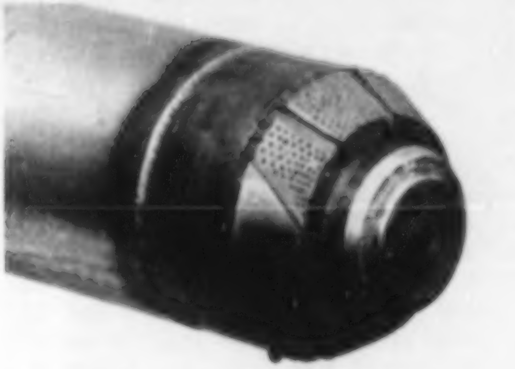
The bit design allows for a multiplicity of external configurations, diamond settings, etc., just as with the wireline bit. The two basic differences between it and a conventional wireline bit are (1) in the manner of fastening to the core barrel outer tube assembly and (2) in the omission of part of the circumferential surface in the area that abuts the driving lugs.

The novel configuration of the bit allows it to be passed through the drill rods. The bit is removed from the drilling position and rotated in two planes, allowing its passage through the inside diameter of the drill rods. However, when the bit is in the drilling position, the hole and core are still cut in a conventional manner.

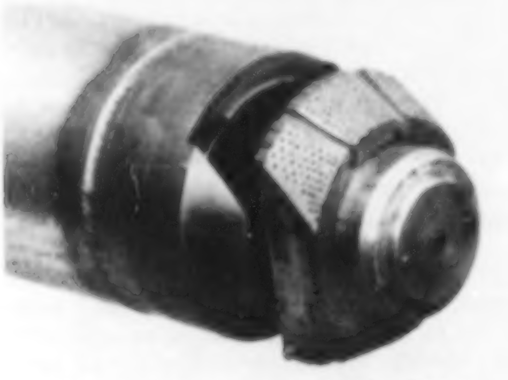
So far as the drilling operation is concerned, the modified wireline core barrel (fig. 4) employed in the retractable bit system operates exactly as conventional wireline core barrel. The basic modification is represented by the addition of a bit holder. This device is fitted on the lower end of the outer tube; when actuated by the retraction or insertion tool, it releases the bit for withdrawal or locks the new bit in place. It does not interfere with the normal function of the inner tube assembly (core barrel).



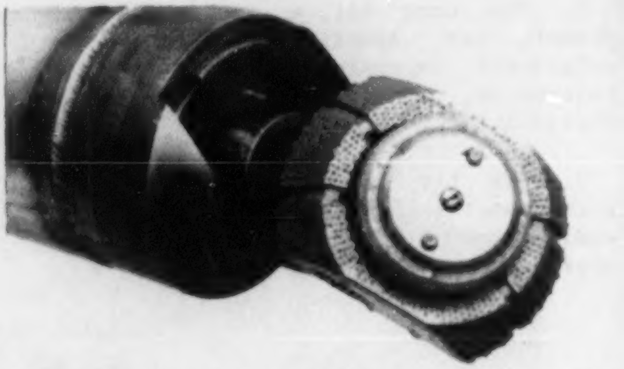
A). Retractable bit after drilling.



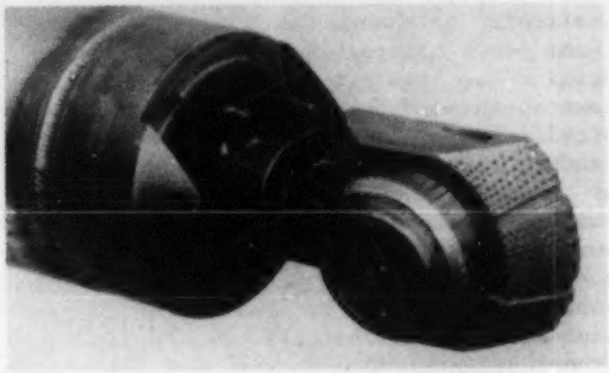
B). Retraction tool engaged, and ready to start retraction process.



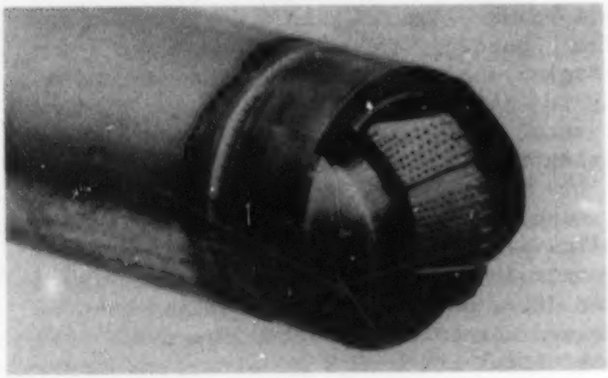
C.) Retraction tool pushing core bit clear of the bit holder.



D). Retractable bit undergoing longitudinal and lateral rotations during retraction cycle.



E). Retractable bit in final orientation at the end of the retraction cycle.



F). Retractable bit being pulled through the drills rods.

FIGURE 2. - Bit retraction sequence starting from the drilling position (A) to the fully retracted position (F).

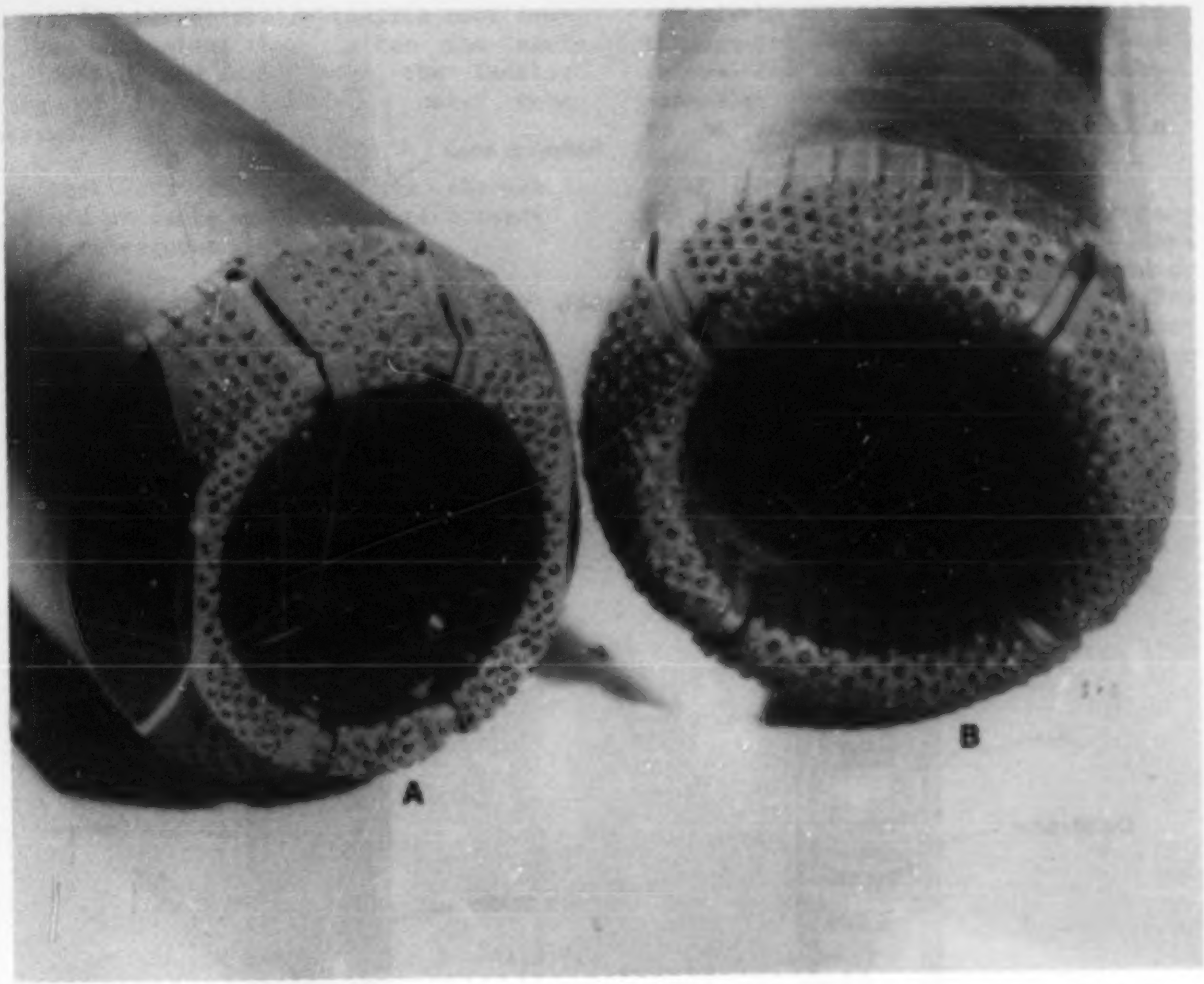


FIGURE 3. - Comparison of the retractable core bit (A) with a conventional wireline core bit (B).

One minor modification to the wireline core barrel is the provision of a groove in the landing ring into which the inserted tool is locked prior to its bit replacement or withdrawing operation. Another difference is a direct connection (no spring) between the inner tube and the spindle bearing.

As a consequence of the retractable bit design, the diameter of the core cut in an N-size is reduced by 1/8 inch (3.2 mm) to 1.75 inch (44.45 mm). This is not considered a serious handicap.

Since the retraction and insertion tools are identical in design and appearance except for one part, their operation will be described as one. The difference is that the direction of motion of the internal drive mechanism is reversed between the tools.

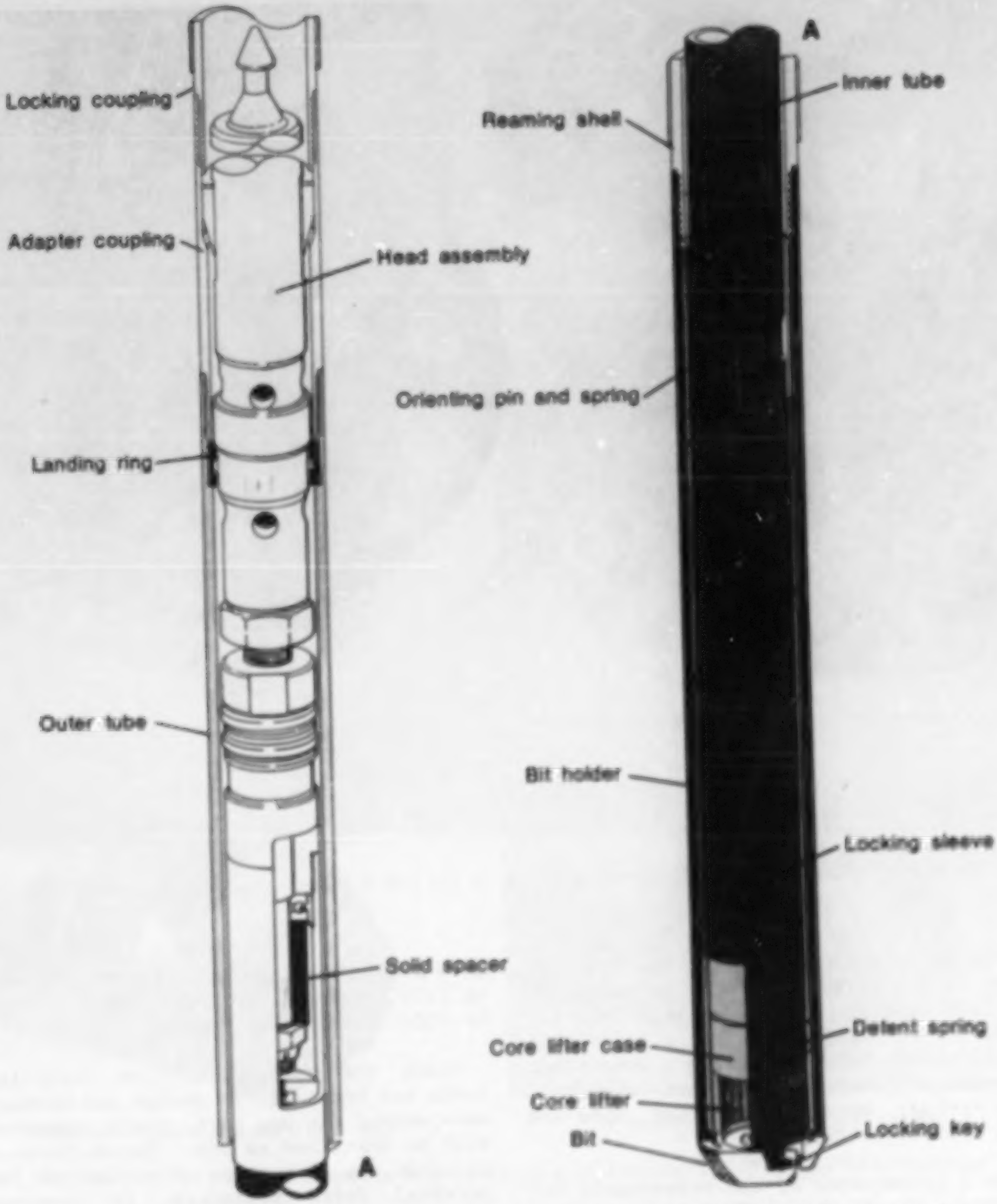

Locking coupling
Adapter coupling
Head assembly
Landing ring
Outer tube
Solid spacer
A
A
Inner tube
Reaming shell
Orienting pin and spring
Bit holder
Locking sleeve
Detent spring
Core lifter case
Core lifter
Bit
Locking key

FIGURE 4. - Schematic of the retractable bit system core barrel. (Unshaded parts are the same as wireline core barrel parts. Medium-shaded parts are changed in dimension only. Dark-shaded parts are the new configurations.)

Simply stated, the retraction tool contains mechanisms that, after the tool is lowered and seats into the landing ring of the core barrel outer tube assembly:

Lock the tool in place through fluid pressure making the tool ready for subsequent operations.

Transform the pulling forces applied by the wireline cable into a series of mechanical actions which grasp the bit; advance it past the end of the bit holder; rotate the bit through two perpendicular planes; and release the tool from the outer tube assembly, enabling the tool and bit to be withdrawn by the wireline cable.

Externally the tools present the appearance of a strong, rigid tube. Few moving parts are exposed. Internally, its function depends on the interaction of a number of sequential parts such as pins, cams, linkages, and drive mechanisms (figs. 5-7).

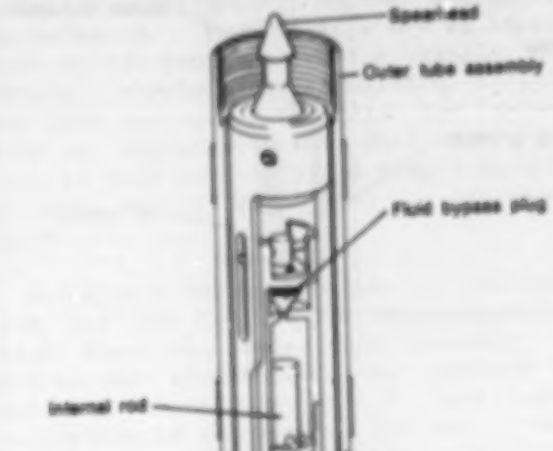

Spearhead
Outer tube assembly
Fluid bypass plug
Internal rod
Piston tube
Camming grooves
Half-dog set screw
Cam sleeve
Tool lock balls
Landing ring
A

FIGURE 5. - Schematic cross section of the upper portion of the retraction tool.

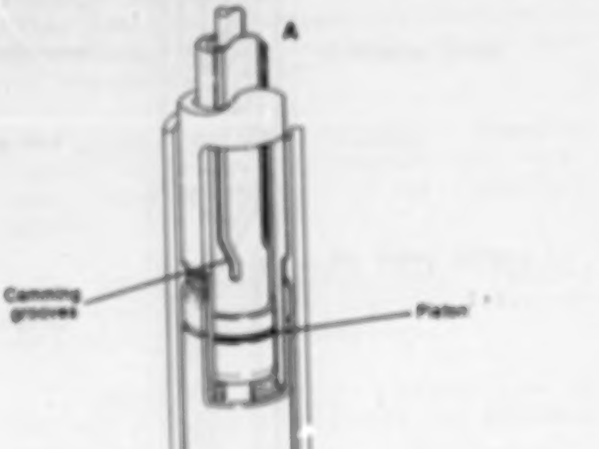

A
Camming grooves
Piston
Internal rod
6 ft (1.8 m) — Extension sections are added here.
Swivel connection
Upper drive screw
B

FIGURE 6. - Schematic cross section of the middle portion of the retraction tool.

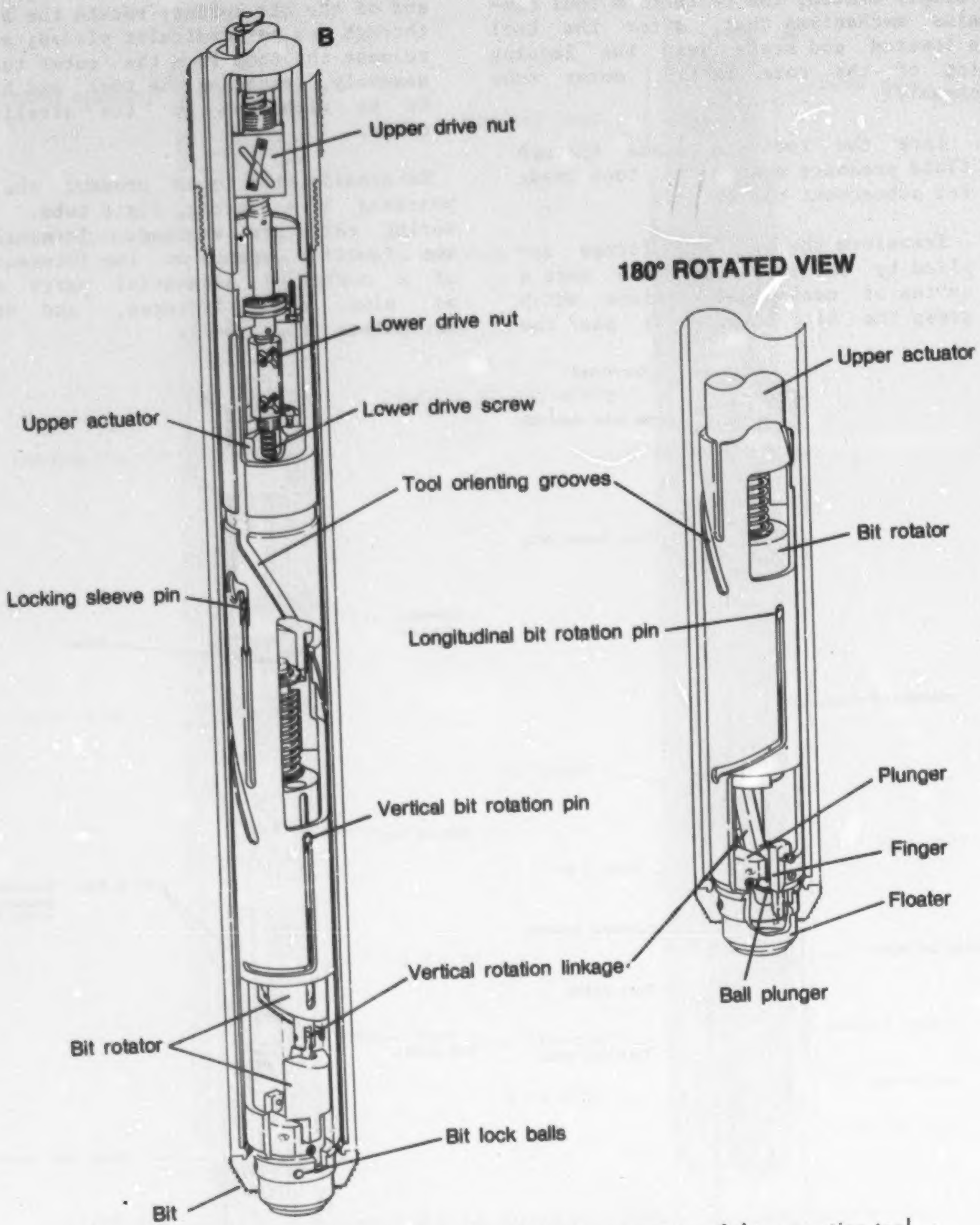

B
Upper drive nut
Lower drive nut
Upper actuator
Lower drive screw
Tool orienting grooves
Locking sleeve pin
Vertical bit rotation pin
Vertical rotation linkage
Bit rotator
Bit lock balls
Bit
180° ROTATED VIEW
Upper actuator
Bit rotator
Longitudinal bit rotation pin
Plunger
Finger
Floater
Ball plunger

FIGURE 7. - Schematic cross section of the lower portion of the retraction tool.

Both tools can accommodate the length of core barrel being used by adding extension sections that fit between the upper and lower portions of the tool (that is, 5-, 10-, 15-foot core barrel). In practice, the procedures currently employed in wireline core drilling apply to the replaceable bit system. Drill setup and collaring of the hole are accomplished as before.

By designing a replaceable bit with its associated retraction-insertion tools to function independently of the inner tube assembly, reliability of the system can be enhanced. This feature was an important design parameter. For example, the driller should be in a position to pull the core bit out of the hole for inspection or replacement only when the decision is made to do so. A simple example of this type of reliability is discussed below.

Assuming a 90% reliability of the system, and that the bit has to be retrieved each time the inner tube assembly is pulled the system could be expected to malfunction one out of 10 core runs, regardless of the footage drilled. However, if the retractable bit were independent of the inner tube assembly (assuming the same reliability), the driller would have to pull the rods out of the hole only once every 10 times that the bit was pulled for inspection as a result of drilling. This concept sounds elementary, yet a historical look at past retractable bit designs and prototypes shows that almost all of the bits functioned as an integral part of the inner tube assembly, which contributed to poor reliability and lack of success.

Benefits of the Replaceable Bit System

A comparison of the time savings of a replaceable bit versus conventional rod pulling indicates that savings are realized where drilling conditions are greater than 1,000 feet (300 m), under normal drilling conditions. However, time savings could be significant in holes less than 1,000 feet if bit life is poor (that is, less than 25 ft/bit). These estimates are considered to be conservative, based on the prototype system. The development of a commercial system could reduce these figures.

In addition to direct time savings through an increase in productivity, other indirect advantages are anticipated from a commercially available replaceable bit system:

1. Less driller fatigue. Changing the bit with the replaceable bit system requires very little physical effort.

2. Lower fuel consumption. Pulling rods requires engine operations that result in high fuel consumption.

3. Improved safety. Pulling rods is the major cause of injuries to drillers and helpers.

4. Improved drilling efficiency. Drillers will tend to change bits to maximize drilling efficiency either because of formation changes or as the bits become dull.

5. Less time required to complete hole. Savings in drilling costs due to faster drilling speeds are only part of the savings. Savings associated with maintaining support personnel and facility onsite could also be significant.

CONCLUSIONS AND CURRENT STATUS

The retractable bit system has undergone field testing under a variety of conditions and rock types. The results of the initial field program are very encouraging and indicate that the retractable bit system is feasible and reliable. An extensive field program by the Longyear Co. is currently underway, as well as engineering evaluation studies to determine the overall commercial feasibility of the system. During 1981, field tests will be conducted on contract drill sites under production conditions. Domestic and foreign patents on the system have been applied for by the Longyear Co.

STRUCTURAL DESIGN FOR DEEP SHAFTS IN HARD ROCKS

By Michael J. Beus[1] and Samuel S. M. Chan[2]

INTRODUCTION

Shaft design has traditionally meant specifications of shaft dimensions, hoisting plant and capacity, skip, guide, and headframe design, sinking method, and location. In recent years there has developed a need for shafts of more sophisticated design in both the established mining areas and the newer districts. Deeper and lower grade ore bodies require faster sinking methods, higher tonnage, lower maintenance, and increased ventilation.

Deep mine shafts frequently experience heavy ground pressure, and failure of the rock and/or supporting structures occur both during sinking and throughout the life of the shaft. This requires almost continuous shaft repair and maintenance, exposure to hazardous working conditions, and excessive operating costs.

The Bureau of Mines is conducting research to develop guidelines for shaft design to improve the structural behavior in deep, vein-type metal mines. The primary objectives are--

1. Define the nature, magnitude, and direction of the stresses acting around proposed shaft openings.

2. Determine the structural sensitivity of various shaft designs to changes in applied load, shape, orientation, and support.

3. Conduct prototype and full-scale field studies.

4. Establish design criteria.

Initial studies concentrated on in situ stress measurement and determination of physical properties. These basic data were developed into predictive equations, and an appropriate range of input values for finite-element method analysis (FEM) was established. By comparing strength of the rock and supporting materials, the stability of the shaft may be predicted. Various parameters are considered including shape, orientation, stress ratio, geologic discontinuities, support system, and time to support. An example shows how a hypothetical shaft might be designed for maximum stability. Prototype field tests compared deformation of circular and rectangular one-half scale test shafts, taking into account geologic and construction variables.

SITE DESCRIPTION

The Coeur d'Alene mining district in northern Idaho is being used as a test area. It is typical of a deep-vein mining area experiencing shaft stability problems, and many shaft test sites are available. The district is situated in the Coeur d'Alene Mountains in northern Idaho. It is mountainous with peaks ranging from 6,000 to 7,000 ft and a regional relief of 3,000 to 4,000 ft. The main rock in the district is Precambrian quartzite and argillite with a maximum thickness of 28,000 ft. The main structural feature is the Osburn Fault, which strikes west-northwest and has extensive displacement. The major fold is the Big Creek anticline, south of the Osburn Fault. Most major mines in the district are located near the Osburn Fault or its branches. The mineral deposits occur as steeply dipping, quartz-siderite veins containing silver-bearing tetrahedrite, galena, and sphalerite. Most of the veins are parallel to the bedding or cut the bedding at small angles.

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[2] Professor of Mining Engineering, University of Idaho, Moscow, Idaho; WAE at Spokane Research Center, Bureau of Mines, Spokane, Wash.

The main mining method is horizontal cut-and-fill stoping using conventional drill, blast, and mucking cycles. Hoisting and rail haulage are the major means for personnel and materials transportation. Rock bolting, timbering, and hydraulic sandfilling are the main types of ground support. Shafts in the district extend to more than 3,000 ft below sea level and more than 8,000 ft below the ground surface. Figure 1 shows an idealized cross section of major mines and shafts in the district with respect to sea level and adjacent topography.

STRESS DETERMINATION

Determination of the magnitude and direction of in situ stresses is essential to reliable design studies. Measurement has been made at various sites in the Coeur d'Alene District ranging in depth from 1,200 to 7,000 ft below the ground surface. Table 1 shows the principal stress ratios in decreasing order, with respect to depth. These data show that the stress conditions are not uniform and the principal stress ratio is not a function of depth, because of tectonic activity influencing the regional stress pattern.

TABLE 1. - Principal stress ratios determined in the Coeur d'Alene mining district

Mine	Principal stress ratio (σ_1/σ_3)	Depth of test site, ft
Sunshine.........	3.18	4,800
Lucky Friday.....	3.11	4,250
Silver Summit....	2.80	5,500
Caladay..........	2.65	1,220
Galena...........	1.85	4,000
Star.............	1.78	7,340
Crescent.........	1.25	5,300

The in situ stress information is further reduced to vertical and horizontal components in table 2, shown in order of increasing depth. The ratio between horizontal and vertical stress and maximum and minimum horizontal stresses is also shown. Failure patterns in vertical raise bores in the district illustrate the strong biaxial stress condition. Figure 2 is an example of shear failure in a 5-ft-diam raise bore, resulting from a strongly directional stress acting 90° from the failed zones.

TABLE 2. - Vertical and horizontal stresses, Coeur d'Alene mining district

Test site	Over-burden, ft	Vertical stress σ_v, psi	σ_{h_1}/σ_v	$\sigma_{h_1}/\sigma_{h_2}$
Caladay..	1,200	1,450	0.88	1.56
Galena...	4,000	5,500	2.37	1.36
Lucky Friday..	4,250	4,770	2.00	1.42
Sunshine.	4,800	7,420	.97	1.81
Crescent.	5,300	6,300	1.24	1.25
Silver Summit..	5,500	7,870	1.87	2.73
Star.....	7,340	7,280	1.43	1.52

By statistical analysis of the data from table 2, linear relationships may be derived enabling prediction of vertical and horizontal stresses at any depth. The expression for vertical stress (σ_v) becomes

$$\sigma_v = 435 + 0.952\ h, \tag{1}$$

and that for maximum horizontal stress (σ_{h_1}) is

$$\sigma_{h_1} = 710 + 1.491\ h. \tag{2}$$

All of the data analyzed through 1980 show that--

1. The vertical stress is comparable to what might be expected from a gravity-loaded mass.

2. The horizontal stresses are greater than the vertical.

3. The horizontal stress ratio ranges from 1:1 to almost 3:1.

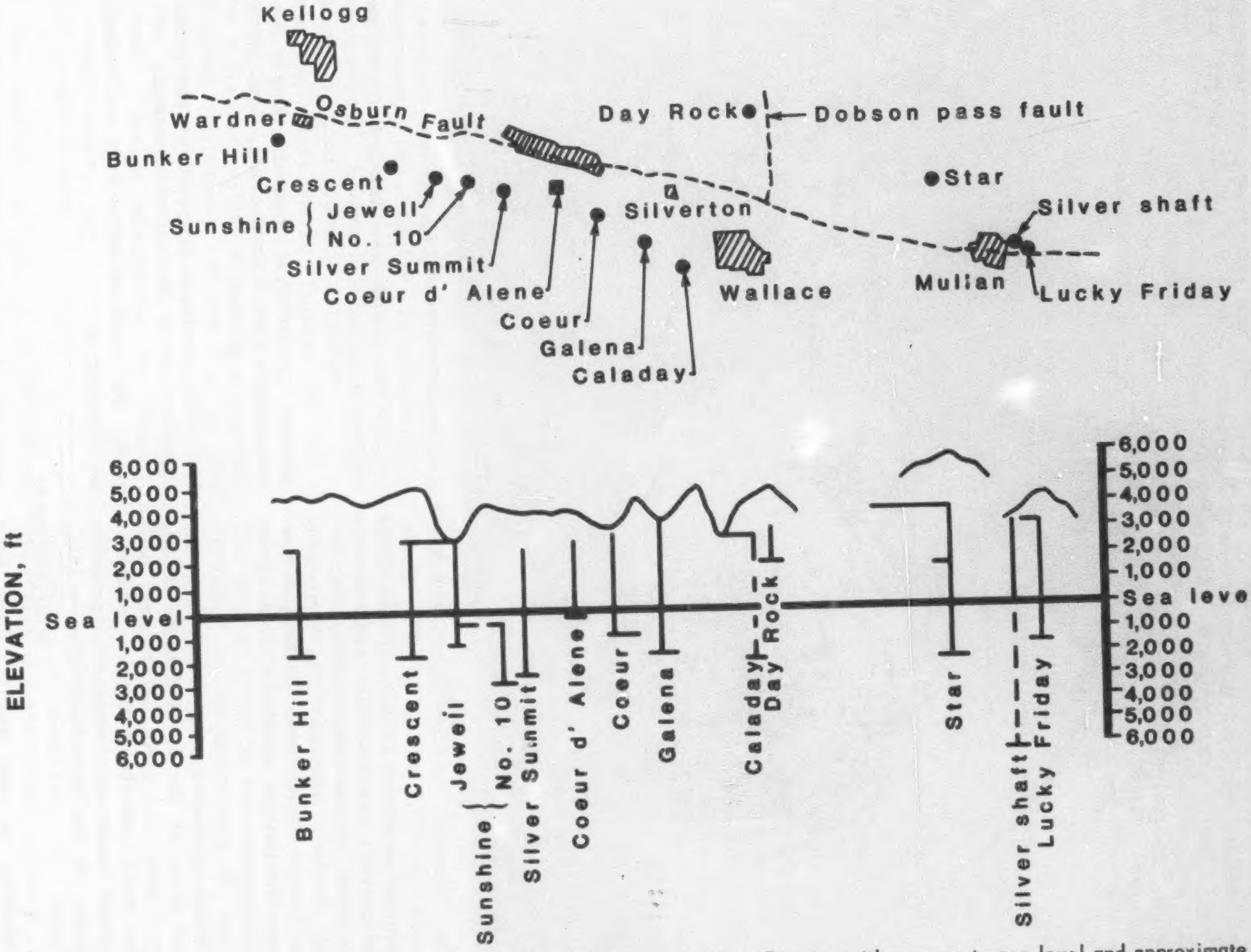
Kellogg
Osburn Fault
Wardner
Bunker Hill
Crescent
Sunshine
Jewell
No. 10
Silver Summit
Coeur d' Alene
Coeur
Galena
Caladay
Day Rock
Dobson pass fault
Silverton
Wallace
Star
Silver shaft
Mullan
Lucky Friday
ELEVATION, ft
6,000
5,000
4,000
3,000
2,000
1,000
Sea level
1,000
2,000
3,000
4,000
5,000
6,000

FIGURE 1. Plan and cross section of major shafts in the Coeur d' Alene District with respect to sea level and approximate topography.

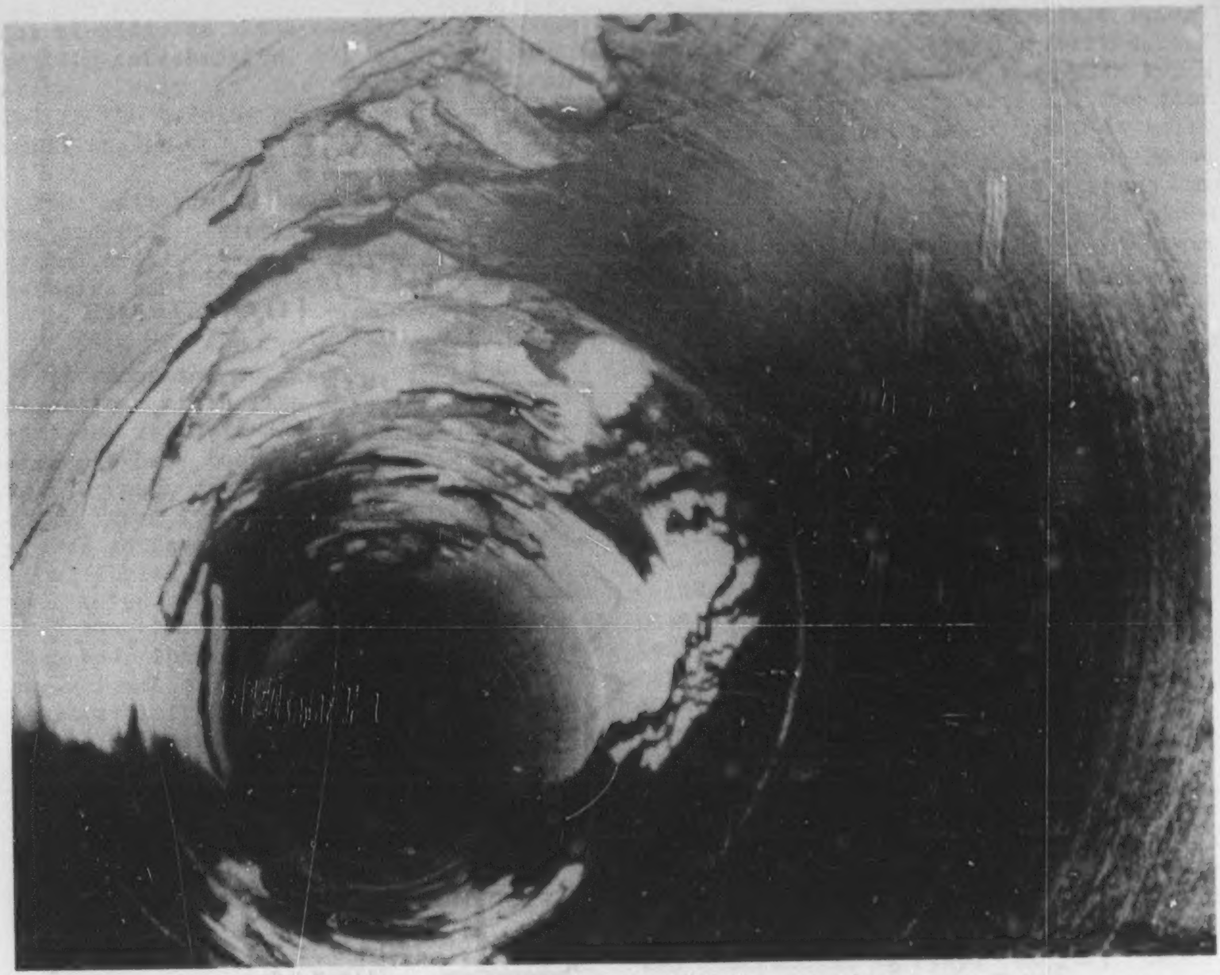

FIGURE 2. - Shear failure in a 5-ft-diam raise bore.

Consideration of the in situ stress field in the overall shaft design procedure is therefore of prime importance, particularly in deep mining. The increasing stress levels could cause stress concentrations around the opening exceeding the rock strength.

Physical properties of rocks from the various mines of the Coeur d'Alene mining district have also been determined, both in situ and in the laboratory. Summarizing these tests on competent quartzite specimens, it has been found that--

1. The rocks are hard, brittle, and deform elastically.

2. The strength of the rocks and rock masses is variable depending upon the mineral content, fracturing, and interlayered argillite.

3. The physical properties vary, particularly with lateral confinement.

4. The rocks creep at very small rates of deformation.

FINITE-ELEMENT ANALYSIS

A variety of FEM models has been developed to illustrate stresses around different shaft shapes and support systems, under nonuniform loading conditions, and for different types of rock behavior. Circular, rectangular, and elliptical cross sections are the three major types of shaft shapes studied. Unsupported and supported shafts with timber, steel, and concrete lining were simulated. The size of the support structure was also varied.

Initial analyses consisted of parametric studies to establish the sensitivity of "variable" and "fixed" conditions. Fixed conditions include applied loads, physical properties, and geologic features; design variables are shape, axis length, orientation, support, and face distance. Table 3 lists physical properties used for the parametric study.

TABLE 3. - Design input parameters for finite-element analysis

Materials	Modulus of elasticity (E), psi	Poisson's ratio, μ	Bulk modulus (B), psi	Shear modulus (G), psi
Hard quartzite..........	10,000,000	0.25	6,666,700	4,000,000
Argillite...............	1,000,000	.25	666,670	333,340
3,000-psi concrete......	3,140,000	.15	1,495,240	1,365,220
9,000-psi concrete......	5,407,500	.15	2,575,000	2,351,000
Carbon steel............	30,000,000	.3	25,000,000	11,538,461
Douglas-fir timber......	1,900,000	.25	1,260,000	760,000
Sand....................	2,000	.15	952	870

Support loads occur as a result of changes in stress state and rock properties and deformation of the shaft wall rock associated with face advance. Either a horizontal or a vertical cross section is possible with a two-dimensional analysis. A vertical plane through the shaft centerline permits analysis of the effects of face advance. A horizontal section permits study of the shape of the cross section, orientation, and support, assuming the support is installed concurrently with excavation. Obviously, this results in considerably higher stresses than would actually be encountered; however, for comparative purposes, this approach is valid.

A two-dimensional, 14-ft-diam, circular shaft model illustrates the simple case of variable applied stress in lined and unlined conditions. Figure 3 shows the influence of the applied stress ratio and the effect of a 2-ft-thick concrete liner. Design procedures for circular shafts in an elastic medium are fairly straightforward, and further analysis is not presented here.

The remainder of this paper emphasizes rectangular shafts, as this is where design procedures and analysis are lacking. The primary structural consideration for rectangular shafts is orientation, with respect both to the prevailing stress field and to discontinuities in the rock mass.

The most stable orientation for a rectangular-shaped opening in a biaxial (2:1 or 3:1) stress field is where the long axis is parallel to the direction of the largest stress. The least stable situation is unidirectional loading, regardless of orientation, or with major stress normal to the longwall of the shaft. Loads up to 9,000 psi have been applied to shaft sections at various ratios and orientation. The preference for orientation remains the same, except that, for increasing stress levels, orientation becomes less a factor as the yield zone increases around the opening. The general trend is towards an increase in yield as the major axis of the shaft is rotated from parallel to perpendicular to the major stress.

Geologic discontinuities such as bedding planes, joints, and/or faults also have a significant effect on shaft stability. A condition of interbedded quartzite and argillite, striking parallel with the long axis of a rectangular shaft, is shown in figure 4. The tangential stress and axial displacement are

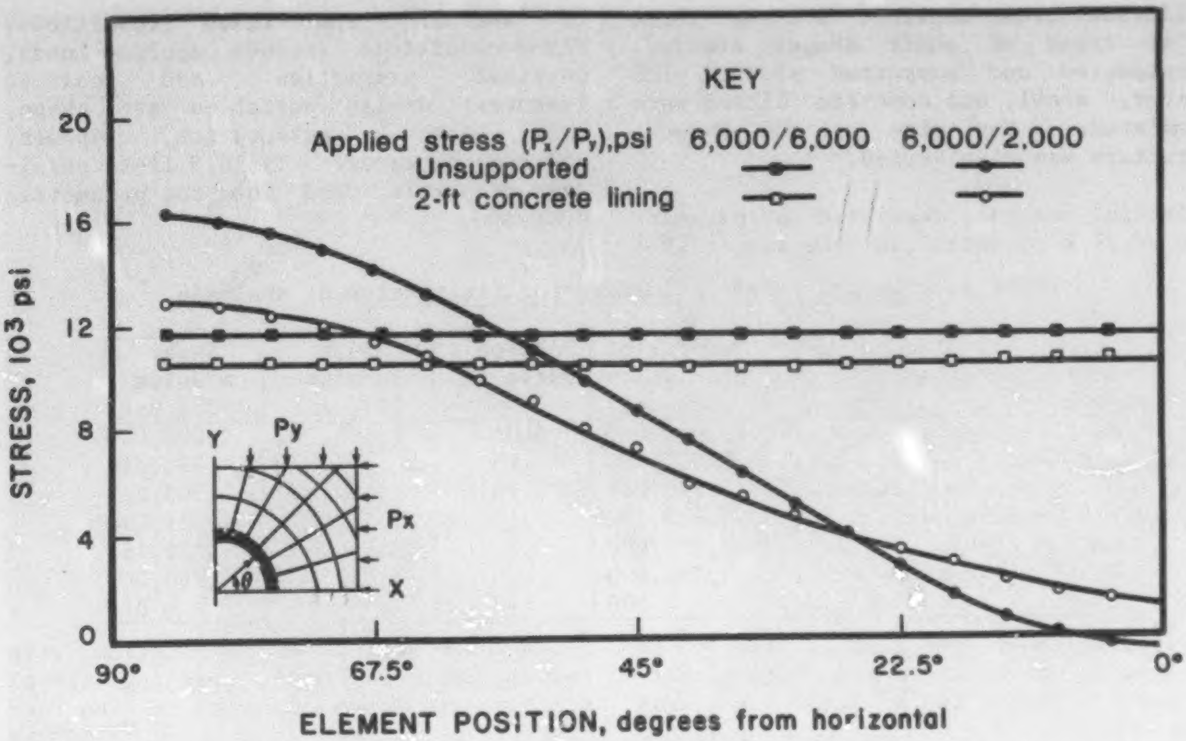

FIGURE 3. - Tangential stress distribution in rocks around a 14-ft-ID supported and unsupported shaft.

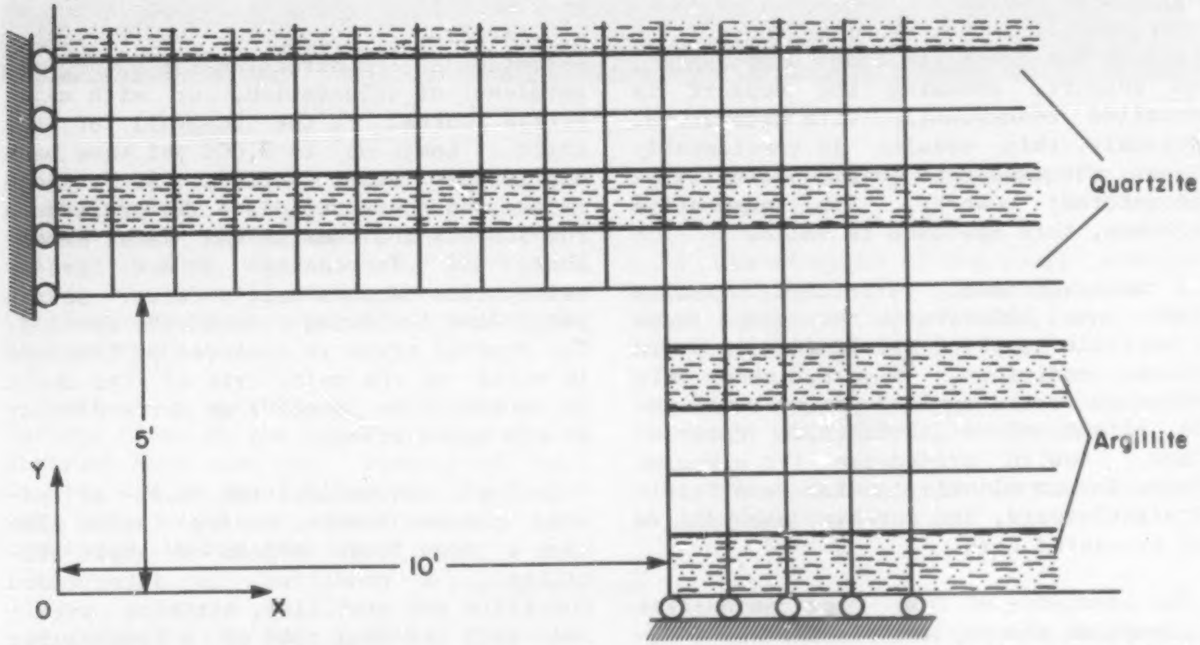
Quartzite
Argillite
5'
10'
Y
O
X

FIGURE 4. - Model of interbedded quartzite and argillite striking parallel with the long axis of the shaft.

considerably greater for the interbedded quartzite and argillite than for quartzite alone. The most extensive displacement occurs where geologic discontinuities strike parallel to the major shaft axis.

The best shaft orientation is to orient the long axis of the shaft parallel with the major stress and perpendicular to bedding as predominant joint patterns in the horizontal plane. Obviously, this can be accomplished only if the strike of the bedding and joints is normal to the major stress. Ultimately, it is envisioned that the best orientation will be a compromise between consideration of the prevailing stress field and of geologic discontinuities.

Additional studies evaluated support systems commonly installed in shafts. Timber, steel, or concrete act as a structural framework to mount the shaft conveyances and provide protection from loose blocks of falling ground. There is little change in stress concentration factor in the rock as a result of installation of these support systems, as illustrated by figure 5. The steel and timber supports result in local high stress zones, particularly at blocking points. The concrete liner results in a more uniform stress distribution around the opening. For timber and steel supports, the highest stress occurs at the dividers. The steel set with timber blocking is very effective in reducing stress concentration radial displacement in the rock.

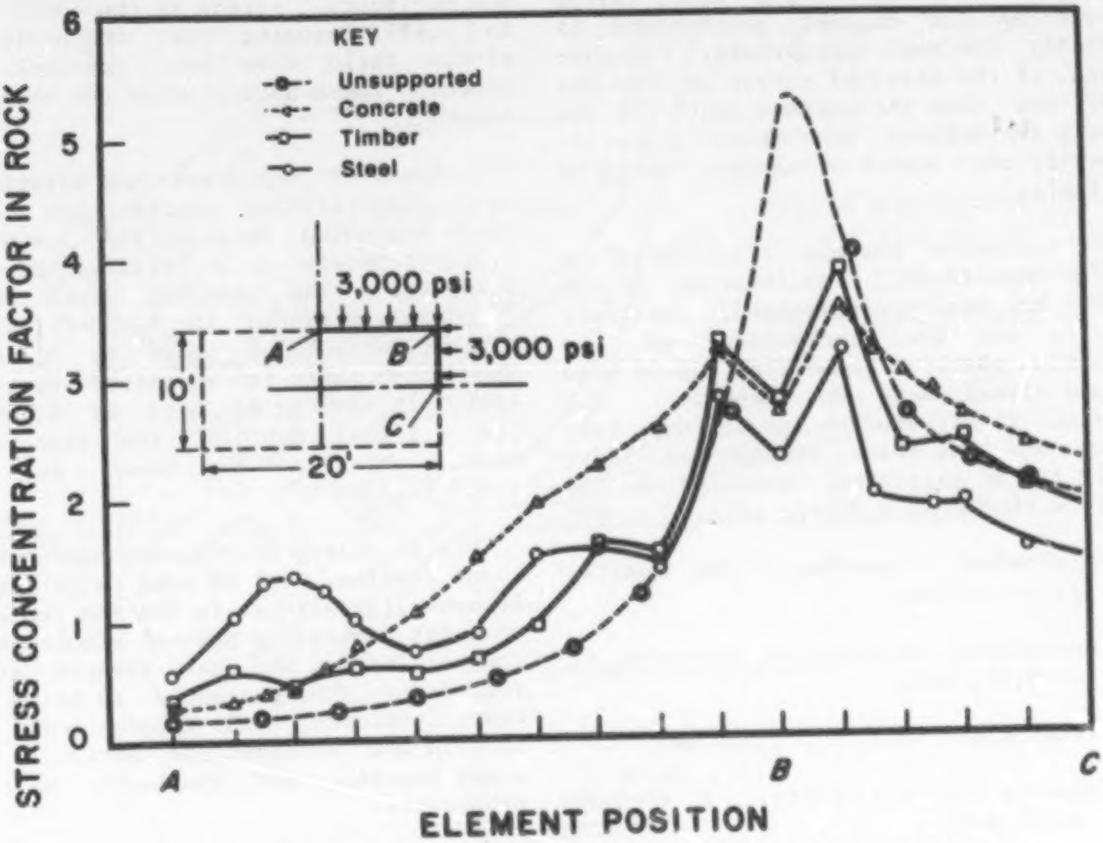

FIGURE 5. - Comparison of supported and unsupported rectangular shafts.

DESIGN ILLUSTRATION

Finite-element analysis may be applied to investigate a wide range of shaft design parameters, and it is still the most economical approach to structural analysis. However, it would be impractical to analyze all the various design combinations. To permit design decisions based on structural performance for a given set of input conditions, the term "critical depth" is suggested. This term has been previously used by Trollope to define the maximum depth that a circular opening would be stable in elastic ground under uniform loading. In the context used here, it is the minimum depth at which material first yields, according to the Mohr-Coulomb yield criteria. For deep shafts, this concept is especially appropriate, since the ultimate depth of mining is undefined and the shaft design permitting the deepest penetration is probably the most appropriate. In contrast, if the vertical extent of the ore body, and thus the maximum depth of the shaft, is defined, such as in a massive deposit, any number of designs would be suitable.

The following example illustrates the design procedure. The location of the shaft has been predetermined, the shaft station has been excavated, and the essential physical properties and in situ ground stress have been measured. The project is situated in mountainous terrain, and the shaft station is under 1,220 ft of quartzite, connected to the surface through a 5,000-ft adit.

The physical properties of the quartzite are as follows:

unconfined compressive strength, C_o = 18,275 psi,

tensile strength, T_o, 1,754 psi,

modulus of elasticity, $E = 9.57 \times 10^6$ psi,

Poisson's ratio, $\mu = 0.205$,

internal friction angle, $\phi = 52.2°$,

cohesion, C = 2,836 psi, and

failure plane angle, $\theta = 60°$.

The in situ stress is

$$\sigma_v = 1,450 \text{ psi}$$

and

$$\sigma_{h_1} = 820 \text{ psi}.$$

From equations 1 and 2, stresses at any depth are estimated. The vertical stress at the 3000 level is 4,452 psi, and the maximum horizontal stress is 7,002 psi. The ratio of $\sigma_{h_1}/\sigma_{h_2}$ is 1.6, so the minimum horizontal stress at the 3000 level is 4,489, assuming that the horizontal stress ratio does not reorient with depth. These data provide the basic FEM input.

Analysis of supported and unsupported 14-ft-diam circular shafts show plastic yield beginning between the 1,400- and 1,500-ft levels at a critical depth of 2,700 ft. The critical depth for a "favorably oriented" 10- by 20-ft, unsupported, rectangular shaft is 4,770 ft. Where the shaft is supported by timber sets, it can go as deep as 4,920 ft. The critical depth of the same shaft when supported by steel sets is 6,820 ft.

This is only a preliminary approach and these results must be used with caution because limitations in the FEM code and computer capability prevent simulation of unequal loading and shaft advance at the same time. The procedure is being further developed, and future work will incorporate construction variables, unequal loading, and nonelastic material properties.

ONE-HALF-SCALE TEST SHAFTS

Mathematical analysis and model studies are invaluable for assessing design alternatives and the effect of various input parameters. FEM studies provide a comparative basis of design parameters. However, the best way to accurately predict the behavior of a full-size opening is by construction and monitoring of a scaled prototype. The actual deformational behavior of circular and rectangular test shafts has been compared by in situ testing. The effect of the construction method and geologic defects, difficult to assess by mathematical or physical modeling, were determined. The experiment simulated, as closely as possible, the excavation of an actual shaft.

Circular and rectangular shaft openings of similar cross-sectional area were excavated at the 1,200-ft depth in an existing shaft station. An 8-ft-diam circular shaft and a 5- by 10-ft rectangular shaft were sunk to a depth of about 27 ft (fig. 6). A full-size shaft at the same orientation, with the long axis normal to the strike of the bedding, was proposed at the site of the rectangular test shaft. Figure 7 is a schematic of the proposed test shaft excavation sequence for both shafts.

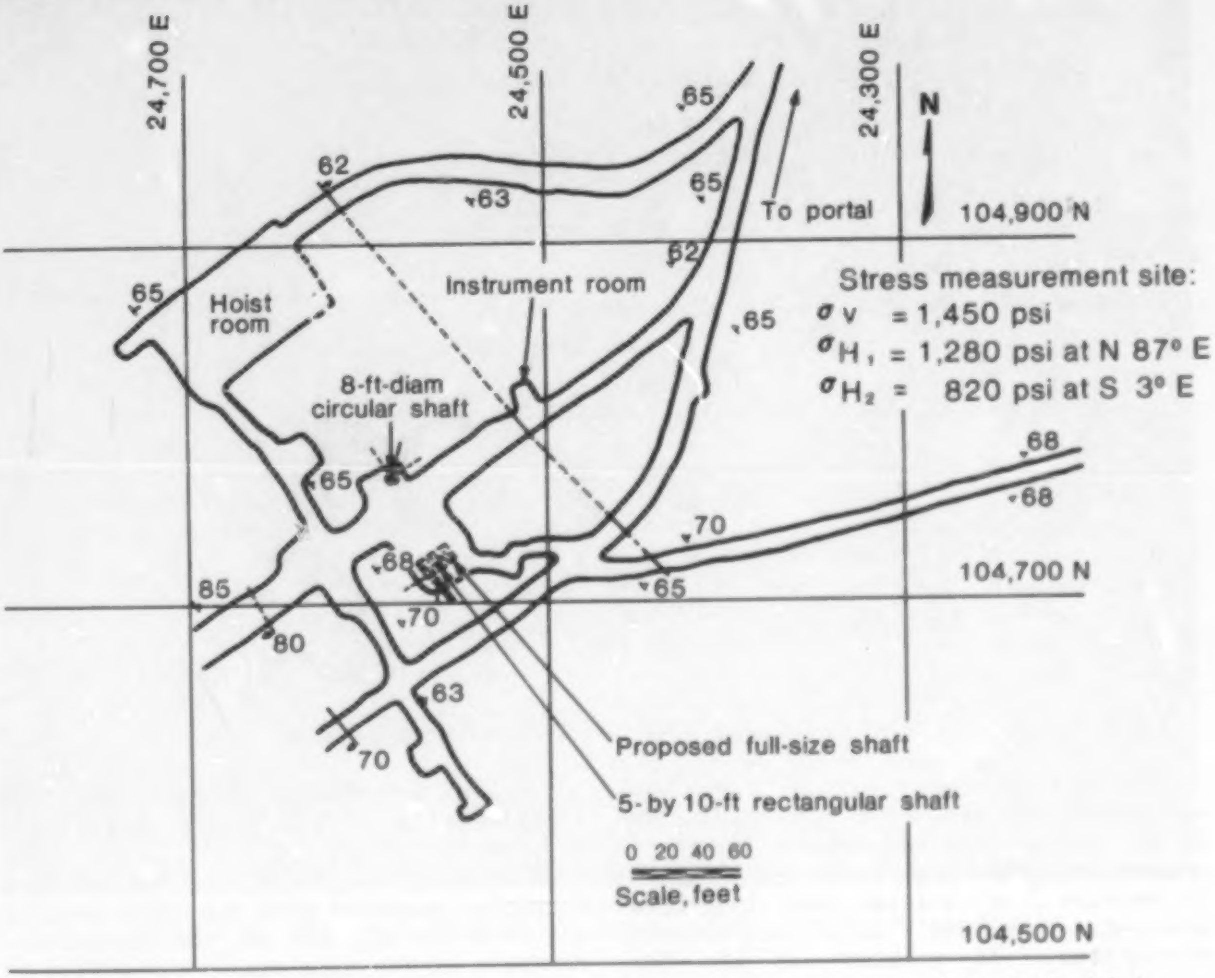
24,700 E
24,500 E
24,300 E
N
65
62
63
65
To portal
104,900 N
62
65
Hoist room
Instrument room
65
Stress measurement site:
σV = 1,450 psi
σH₁ = 1,280 psi at N 87° E
σH₂ = 820 psi at S 3° E
8-ft-diam circular shaft
65
68
68
70
68
85
65
104,700 N
70
80
63
70
Proposed full-size shaft
5- by 10-ft rectangular shaft
0 20 40 60
Scale, feet
104,500 N

FIGURE 6. - Plan view of proposed test shafts, hoist room, and connecting drifts in the Caladay Project.

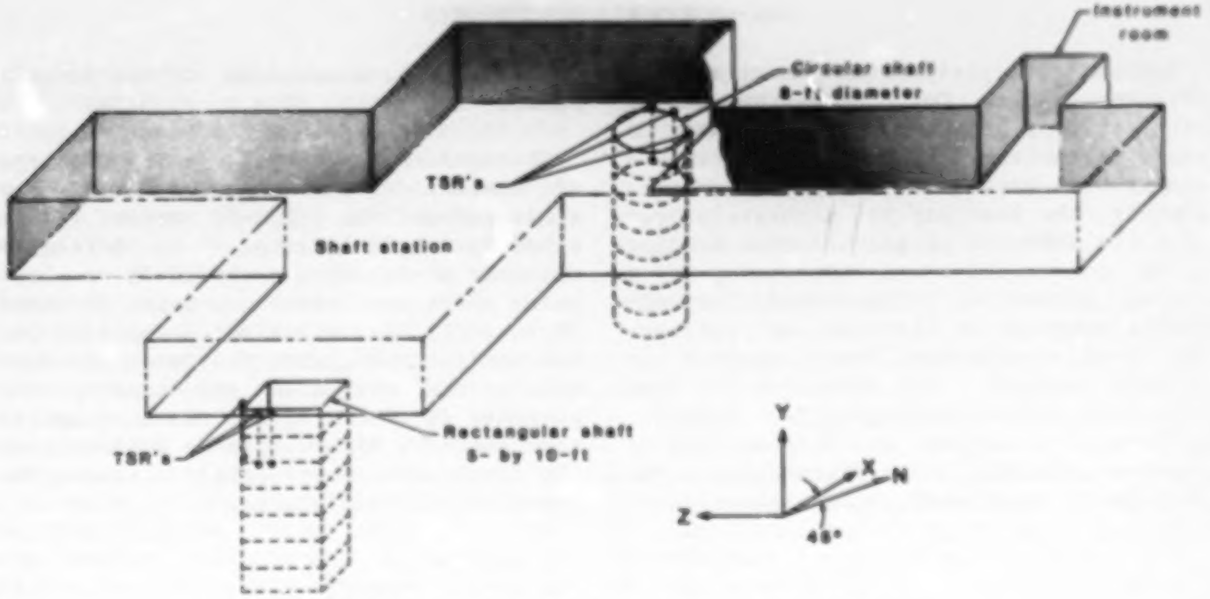
Instrument room
Circular shaft 8-ft diameter
TSR's
Shaft station
TSR's
Rectangular shaft 5- by 10-ft
Y
X
N
Z
48°

FIGURE 7. - Schematic of the station and test shafts.

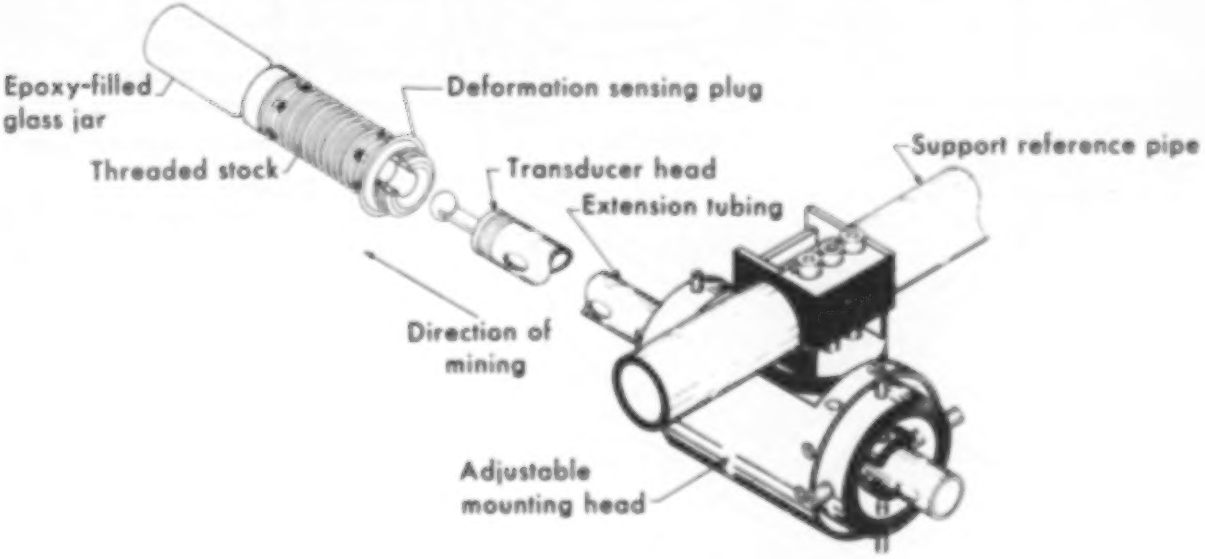
Epoxy-filled glass jar
Deformation sensing plug
Threaded stock
Transducer head
Extension tubing
Support reference pipe
Direction of mining
Adjustable mounting head

FIGURE 8. - Components of the TSR measurement system.

The instrument used in the test was developed and patented by the Bureau in 1971. It is called a tunnel stress relaxation (TSR) gage and is designed to measure the initial and long-term deformation around large underground excavations. It consists of the TSR probe, a deformation sensor plug, an adjustable mounting head, and an anchored support reference pipe (fig. 8). The sensing plug is epoxied into the end of a borehole parallel with the mine opening and fits the ball end of the transducer. Movement of the plug causes deflection of the probe. Figure 9 shows the instruments installed around the completed circular shaft. The data acquisition station was located in an instrumentation room downdrift from the shafts. Blasting was controlled from this room, and detonation was initiated with the start of the data acquisition cycle to obtain maximum data.

After initializing the instruments, the round was detonated, readings allowed to stabilize, and the process repeated. Shaft muck was removed with a backhoe and slusher-clam bucket and loaded into a train loader for transport to the surface. Final depth of the circular shaft after eight rounds, over a period of about 2 weeks, was 24 ft, roughly two opening diameters (16 ft) beyond the TSR sensing plane (fig. 9). The rectangular shaft was excavated with seven consecutive rounds to a final approximate depth of 26 ft, roughly 17 ft beyond the TSR sensing plane (fig. 10).

FIGURE 9. - The completed circular test shaft and the installed instrumentation.

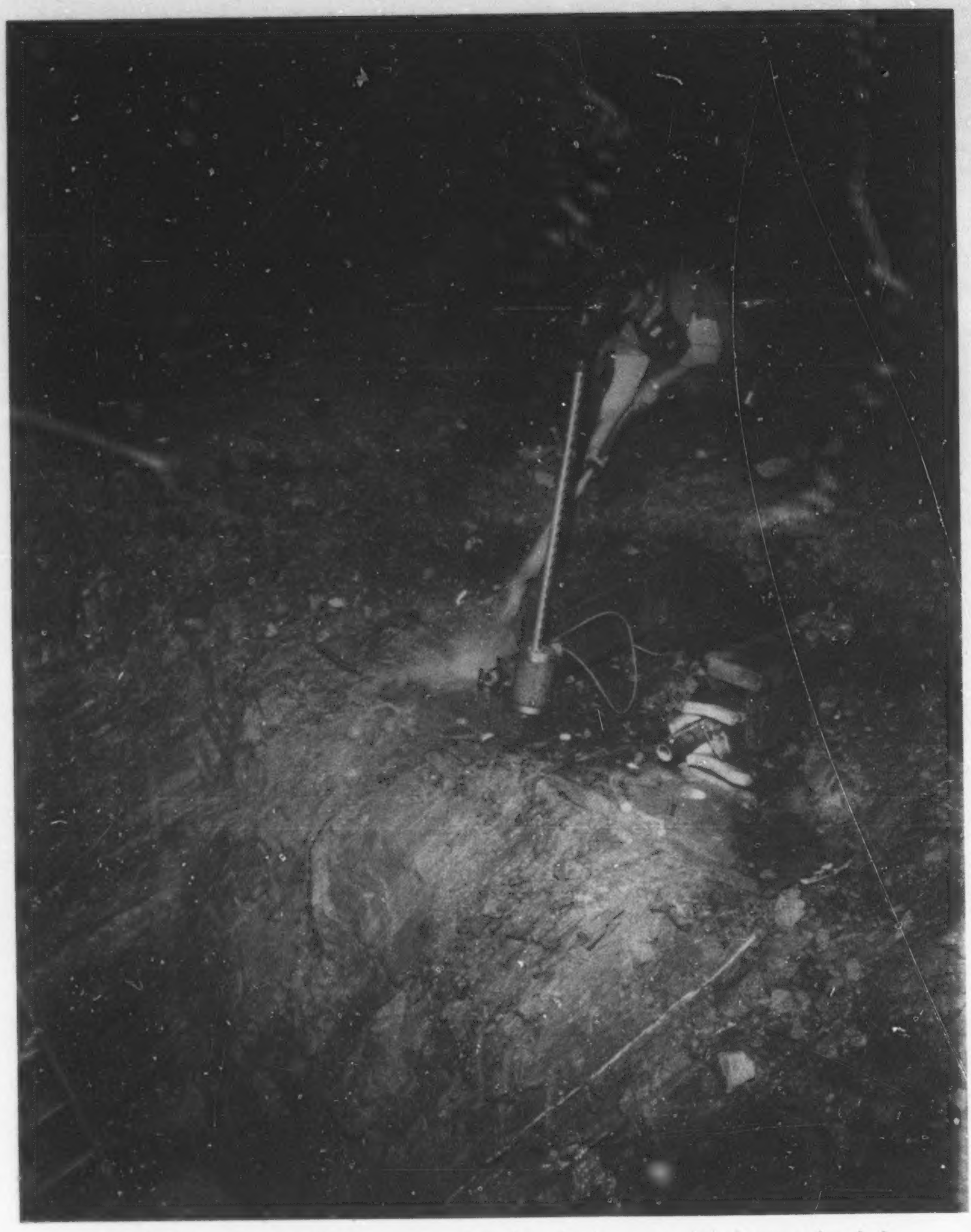

FIGURE 10. - The installed instrumentation along the long axis of the rectangular shaft.

Each blast developed a characteristic pattern, or trend, during the blast. Rock response could be classified as either elastic or inelastic. A relatively slow and small deformation occurred for about 1 hr following each blast at a slowly decreasing rate. Figures 11 and 12 show the displacement traces for the circular and rectangular shafts, respectively.

Considerable difficulty was experienced with excessive water and the inherent complications resulting from the use of explosives and their effect on the instrumentation. Despite these difficulties, valid data were obtained, and some significant data trends were noted. Maximum displacement occurred normal to the major geologic planes of weakness (bedding planes), and in the case of the rectangular shaft, was extreme to the point of endwall failure. Elastic instability was evident as beam bending at the midpoint of the long axis of the rectangular shaft. Displacements generally ceased as the face advanced two-opening diameters (about 17 ft) beyond the TSR sensing plane. A 4-month monitoring period following completion of the openings showed no noticeable creep characteristics. The smallest displacements were measured at the 0° and 45° positions in the circular shaft (parallel and 45° to the bedding) and between the center and the corner of the long axis of the rectangular shaft.

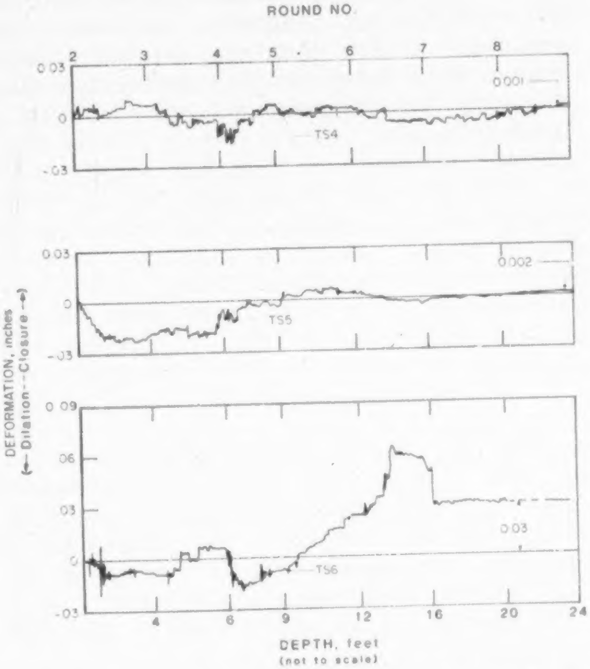

FIGURE 11. - Deformation around the circular test shaft measured with the TSR's as a function of shaft depth.

Comparing the actual data record for the circular and rectangular test shaft, the following conclusions regarding their relative stability are appropriate.

1. The circular shaft was less sensitive to the excavation process, as shown by the subdued nature of the deformation record.

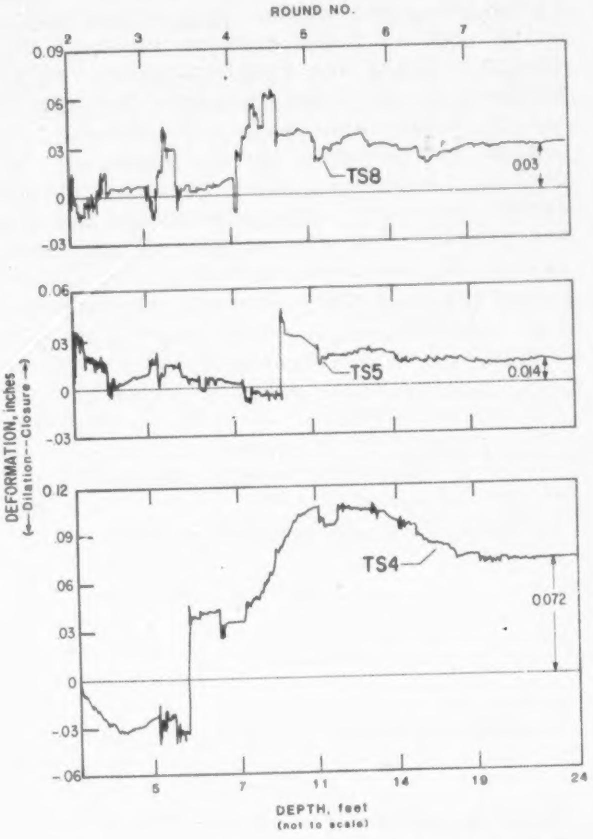

FIGURE 12. - Deformation around the rectangular test shaft measured with the TSR's as a function of shaft depth.

2. The magnitude of displacement was generally smaller and more uniform around the circular shaft and less affected by geologic discontinuities.

3. The rectangular shape is more susceptible to the effects of elastic instability; that is, beam bending and buckling, and geologic discontinuities.

SUMMARY

This research centers around the controversy of circular versus rectangular shaft shapes. Stress and physical property measurements have shown that the horizontal in situ stress around a shaft can be several times larger than the vertical stress. The horizontal stresses are unequal, with up to a 3:1 ratio between major and minor stresses. These data justify detailed investigations of various shaft designs to develop long-term stability of the opening.

With the FEM technique, the structural implications of various design and construction variables are investigated using actual field data as input. Many design options are available that satisfy stress concentration criteria for a given set of input conditions. To illustrate a design procedure that minimizes the design variables, the term "critical depth" is used. This concept is based on the premise that the single "best" shaft design is one that permits the deepest penetration of the deposit, an approach unique to a steeply dipping, vein-type ore body. Selection is based on structural performance in plastic ground conditions with unequal horizontal loads.

It is difficult to mathematically model actual construction practices and geologic features. Field testing of scaled test shafts has allowed investigation of these parameters for circular and rectangular shaft shapes.

The FEM approach is being improved to more realistically model actual construction practices and rock structure. A new computer at the Bureau's Spokane Research Center and modified FEM codes will permit improved structural analysis. In addition, full-sized circular and rectangular shafts are now being instrumented to fully describe the structural behavior of the shaft and support system. An improved shaft design method should emerge from these studies as further data are obtained and compared with currently used criteria.

BOREHOLE DEVIATION CONTROL

By E. H. Skinner[1] and N. P. Callas[2]

INTRODUCTION

This paper presents the results of contract research for the past 2 years at the Spokane Research Center (2). Two areas of borehole research have been undertaken: (1) the mathematic analysis for the preferred method of calculating borehole surveys, and (2) the 2-D analysis of drill-string mechanics. The text of this paper will briefly summarize the results of each of these areas. The Hewlett-Packard (HP) and Texas Instruments (TI) computer programs for the borehole survey calculation method are available upon request from the Spokane Research Center. An appendix briefly covers the elementary theory of drill-string mechanics.

It was from the review of the literature that our research plan developed. It was noted that many papers addressed problems of borehole surveys. It is well acknowledged that mining applications of drilling require much more precision in drilling and surveying than most other drilling, such as oil field-type drilling. Another problem noted in the literature was the drilling of straight holes to specific underground targets and the problems of drill-pipe twistoffs, particularly at the bit end and at stabilizers that were placed near the drill bit. We believe that we have now addressed each of these problems to the limits of present technology (5).

Straight-hole drilling may use intentionally deviated drill holes to help achieve optimal drill-hole usage and, particularly, will extend the capability of surface drilling equipment. In many cases, carefully engineered directional drilling offers the only method for reaching desired target locations. However, as the borehole inclination angle increases, peculiar drilling problems begin to develop that severely limit the horizontal extent of directional drilling. Among these problems are (1) a general tendency to lose the ability to control bit weight and provide bit guidance; (2) optimum rate of penetration is lost and, therefore, economic rig use is lost; (3) the ability to clean the drill hole is decreased and maintaining effective drilling hydraulics becomes a problem; (4) drill pipe sticking in the borehole; and (5) general difficulties in downhole movement of drill pipe and casing, and in wireline tools. The advantages in being able to drill to great horizontal distances at controlled angles is that the effective drilling area may be increased by as much as 10 times. Certainly, the technique could find application in a number of special drilling applications.

A list of some of the possible mining uses for straight-hole drilling, controlled directional drilling, as well as general mine drilling applications, is as follows:

1. Geologic data collection.
2. Ore-body definition.
3. Ore reserve analysis.
4. Surface environmental considerations.
5. Hazard detection and/or evasion drilling.
6. Health and safety emergency exits.
7. Water inflow and shutoff control methods.

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8. Closely controlled vertical and inclined shaft predrilling.

9. Ore pass development.

10. Drilling for ventilation openings.

11. Manway raises and stope raise development.

12. Large-scale blasthole mining methods.

13. Boundary drilling for bulk mining methods.

14. Drilling for induced ore caving.

15. Special drill holes for running utilities.

16. Special drill holes for waste backfill.

17. Special drill holes for hydraulic pumping of ore.

18. Special drill holes for firefighting during a mine disaster.

19. Multiple drill holes from one central drilling location.

20. Drilling for the promising new method of solution mining.

Each of these special drill-hole programs will require different engineering criteria. To illustrate one special application of straight-hole drilling with innovative mine development consequences is the following ore pass development exclusively through raise-boreholes using straight-hole pilot-hole drilling practices. This example (fig. 1) proposes use of 400-ft level development instead of the customary 200-ft level development; thus, we eliminate one level from the present system, which will allow 100% recovery of ore from one sill and vast savings in time and cost for level development and cutting shaft stations. Of course, ore-body geometry and many other considerations may not allow total development of this concept at every mine.

However, some of the advantages are the following:

1. Eliminates one 200-ft level development with time and cost savings for all development expense.

2. Removes one less rock-burst-prone sill mining sequence.

3. Recovers all ore in one 40-ft sill interval now lost with the 200-ft level.

4. Enables time and cost savings in station cutting and drift development for one level towards the vein. Also eliminates a tremendous amount of waste muck removal from the extra 200-ft level.

5. Consolidates hoisting from 400-ft levels rather than from 200-ft levels, which may be better programed for automatic hoisting control from fewer levels.

6. The circular ore pass with an inclination of 50° to 80°, and a diameter of 5 ft or more is a preferred opening for an ore pass.

7. Savings in level timber consumption and other supplies for the 200-ft level development.

8. Raise-bore cuttings are ideal for backfill use and roadway surfaces.

The disadvantages are--

1. Requires accurate pilot hole drilling.

2. May require a lined muck pass design.

3. Raise-bore holes totaling more than 1,000 ft may be seen as time consuming to complete.

4. May experience muck hangups in long ore passes.

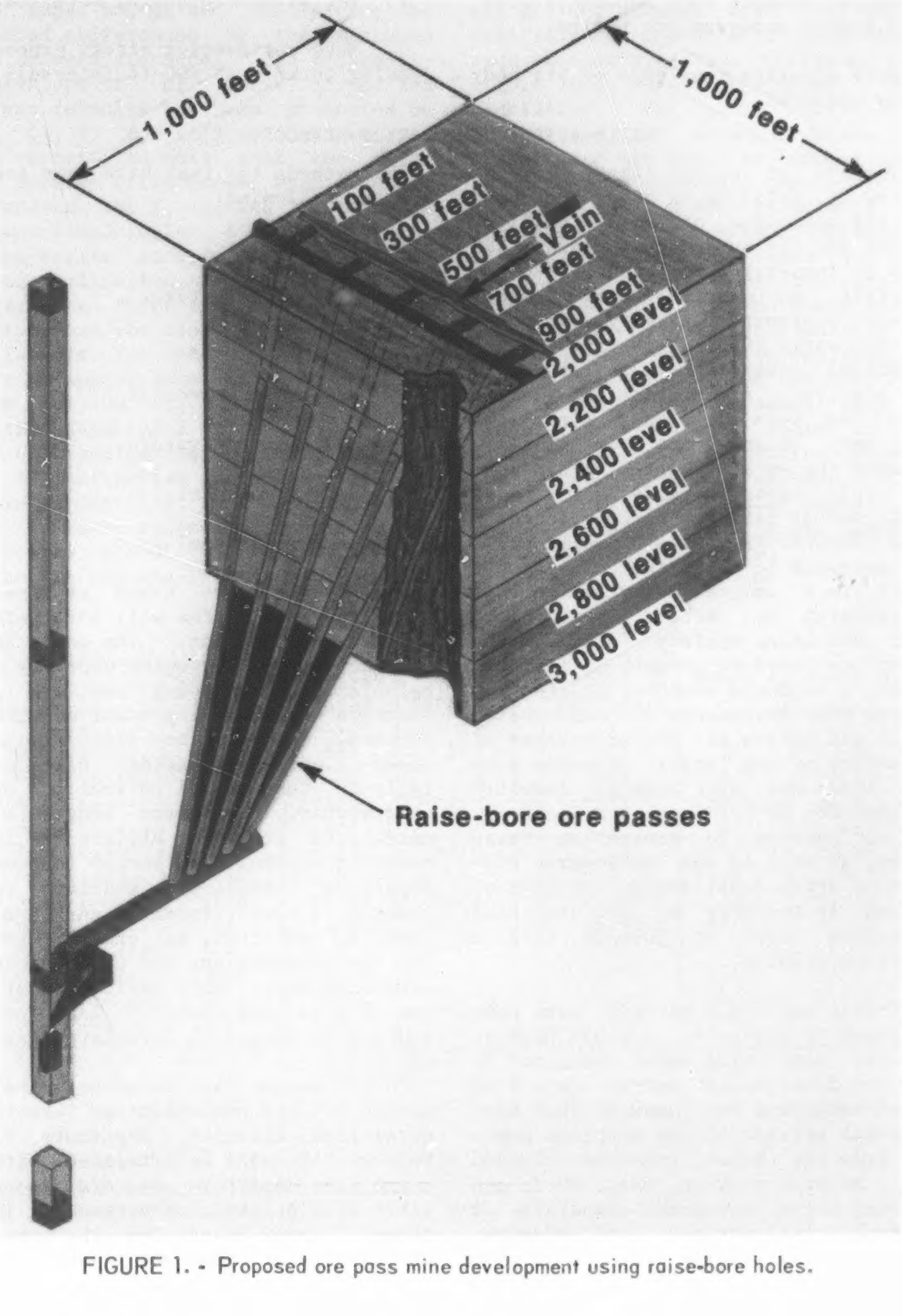
1,000 feet
1,000 feet
100 feet
300 feet
500 feet
Vein
700 feet
900 feet
2,000 level
2,200 level
2,400 level
2,600 level
2,800 level
3,000 level
Raise-bore ore passes

FIGURE 1. - Proposed ore pass mine development using raise-bore holes.

5. May experience rock mechanics problems in highly stressed ore bodies.

6. More accurate knowledge of ore-body geometry required.

7. Better mine planning required.

8. More engineering effort expended in loading pockets at 400-ft intervals.

DRILL SURVEY CALCULATION METHODS

Instruments of various kinds for the surveying of drill holes have been used since the beginning of modern drilling and today are the subject of massive R&D efforts by industry brought on by the energy crisis. An associated subject area, which was apparently largely overlooked until the early 1960's, is the "best" mathematical presentation of borehole survey results. A profusion of various methods has been developed since the 1960's, many of which are very similar, and indeed the work presented herein considers only seven methods. The most accurate method has been documented as a Fortran computer program and implements a vector approach to the data. The vector approach is a unique contribution from this research and offers calculations free of the usual anxiety in arithmetic calculations based on azimuth angles. In addition, a handheld computer program was developed that implements the same vector approach and offers all the advantages of computations of the larger computer program. These are available to industry for either the HP-67/97 or the TI-58/59. It is our purpose in presenting these programs, as well as the subsequent discussion on drill stabilizers, to present the means to industry to perform these calculations right at jobsite with a handheld calculator.

Drill-hole surveying methods have come to be known as two-point methods because successive departures are computed by using the directional survey data from adjacent stations to compute the next incremental horizontal and vertical positions from the known previous station point. At each station, then, there are the usual three measured quantities of elevation, inclination, and azimuth, which are the customary directional survey data associated with that station. Nothing has been changed from the previous collection of survey data.

The methods (4) that have been analyzed are the following:

1. Circular arc.
2. Minimum curvature.
3. Average angle.
4. Balanced tangential.
5. Radius of curvature.
6. Walstrom, model 1.
7. Walstrom, model 2.

Unit vectors are taken as directions along the axis of the well bore at adjacent survey stations. The angle between the vectors is the dogleg severity and is calculated by applying the arc cosine function to the dot product of the unit vectors. Hence, the vector departure along a uniquely defined circular arc falls in the chordal direction of the unit vector. The chord length is also calculated from the difference in the measured depths. The vector increment of departure between two stations can be given by a vector formula; and the final form, so obtained, is exactly the same for the circular arc and the minimum curvature method. This verified that these two models are exactly identical and would give identical departure results.

In the sequel that developed the other models, it is convenient to examine the normalized circular departure vector. This vector will be compared with the normalized departure vectors among the other five distinctive methods. In this manner, upper bounds for the normalized differences of departure results between pairs of models were developed. Absolute differences can then be calculated from these models by merely taking the maximum of the upper bounds and multiplying the normalized differences by the measured length between stations. The mathematical development of these normalized departure formulas has been presented by Callas (4, 9) and will not be reviewed herein except to note that the upper bounds between differences were obtained by expanding the X, Y, and Z terms as two-dimensional Taylor sine and cosine function series in terms of the differences of inclination and azimuth. Incidentally, the first two terms of these expansions can be used as an approximation formula for the dogleg severity angle with appropriate change of units to degrees per 100 ft. Formulas for the normalized departures X, Y, and Z, in terms of these series expansion differences of inclination and azimuth, are given in tables 1, 2, and 3, with coefficients of each expansion tabulated for each of the six distinct models. Note that the 0th and the 1st order terms for all of the methods are identical, thus verifying the approximate equivalency of each two-point method. Different coefficients arise when higher order terms are considered. For small unit values of inclination and azimuth, these higher order terms are small in comparison with the lower order terms. Hence, the differences are also relatively small. Therefore, the comparison between pairs of models reveals that they are exactly equivalent up to, but not including, second-order terms when their formulas are considered as series expansion of the differences of successive inclination and azimuth angles. These results confirm the long-held opinion among many directional surveyors that there is little difference between departures computed by the various two-point methods. Recall, too, that the results also present a simply applied approximation for computation of dogleg severity angle.

TABLE 1. - Normalized departures for X

	Circular arc	Average angle	Balanced tangential	Radius of curvature	Model 4	Model 5
a_0..........	1	1	1	1	1	1
a_{10}.........	1/2	1/2	1/2	1/2	1/2	1/2
a_{01}.........	1/2	1/2	1/2	1/2	1/2	1/2
a_{20}.........	-1/6	-1/8	-1/4	-1/6	-1/6	-1/4
a_{11}.........	1/2	1/4	1/2	1/4	1/3	1/4
a_{02}.........	-1/4	-1/8	-1/4	-1/6	-1/6	-1/4
b_{02}.........	1/12	0	0	0	0	0
a_{30}.........	-1/24	-1/48	-1/12	-1/24	-1/24	-1/12
a_{12}.........	-1/4	-1/16	-1/4	-1/12	-1/8	-1/8
b_{12}.........	1/8	0	0	0	0	0
a_{21}.........	-5/24	-1/16	-1/4	-1/12	-1/8	-1/8
a_{03}.........	-1/12	-1/48	-1/12	-1/24	-1/24	-1/12
b_{03}.........	1/24	0	0	0	0	0

$$
\begin{aligned}
X = {} & a_0 \sin I \sin A + a_{10} \cos I \sin A \, dI + a_{01} \sin I \cos A \, dA \\
& + a_{20} \sin I \sin A \, dI^2 + a_{11} \cos I \cos A \, dI \, dA + a_{02} \sin I \sin A \, dA^2 \\
& + b_{02} \sin^3 I \sin A \, dA^2 + a_{30} \cos I \sin A \, dI^3 \\
& + a_{21} \sin I \cos A \, dI^2 \, dA + a_{12} \cos I \sin A \, dI \, dA^2 \\
& + b_{12} \cos I \sin^2 I \sin A \, dI \, dA^2 + a_{03} \sin I \cos A \, dA^3 \\
& + b_{03} \sin^3 I \cos A \, dA^3 + O_4.
\end{aligned}
$$

TABLE 2. - Normalized departures for Y

	Circular arc	Average angle	Balanced tangential	Radius of curvature	Model 4	Model 5
a_0.........	1	1	1	1	1	1
a_{10}........	1/2	1/2	1/2	1/2	1/2	1/2
$-a_{01}$.......	-1/2	-1/2	-1/2	-1/2	-1/2	-1/2
a_{20}........	-1/6	-1/8	-1/4	-1/6	-1/6	-1/4
$-a_{11}$.......	-1/2	-1/4	-1/2	-1/4	-1/3	-1/4
a_{02}........	-1/4	-1/8	-1/4	-1/6	-1/6	-1/4
b_{02}........	1/12	0	0	0	0	0
a_{30}........	-1/24	-1/48	-1/12	-1/24	-1/24	-1/12
a_{12}........	-1/4	-1/16	-1/4	-1/12	-1/8	-1/8
b_{12}........	1/8	0	0	0	0	0
$-a_{21}$.......	5/24	1/16	1/4	1/12	1/8	1/8
$-a_{03}$.......	1/12	1/48	1/12	1/24	1/24	1/12
$-b_{03}$.......	-1/24	0	0	0	0	0

$$Y = a_0 \sin I \cos A + a_{10} \cos I \cos A \, dI + a_{01} \sin I \sin A \, dA$$
$$+ a_{20} \sin I \cos A \, dI^2 - a_{11} \cos I \sin A \, dI \, dA + a_{02} \sin I \cos A \, dA^2$$
$$+ b_{02} \sin^3 I \cos A \, dA^2 + a_{30} \cos I \cos A \, dI^3$$
$$+ a_{21} \sin I \sin A \, dI^2 \, dA + a_{12} \cos I \cos A \, dI \, dA^2$$
$$+ b_{12} \cos I \sin^2 I \cos A \, dI \, dA^2 - a_{03} \sin I \sin A \, dA^3$$
$$- b_{03} \sin^3 I \sin A \, dA^3 + O_4.$$

TABLE 3. - Normalized departures for Z

	Circular arc	Average angle	Balanced tangential	Radius of curvature	Model 4	Model 5
a_0..........	1	1	1	1	1	1
$-a_{10}$........	-1/2	-1/2	-1/2	-1/2	-1/2	-1/2
a_{20}.........	-1/6	-1/8	-1/4	-1/6	-1/6	-1/4
b_{02}.........	1/12	0	0	0	0	0
$-a_{30}$........	1/24	1/48	1/12	1/24	1/24	1/12
c_{12}.........	-1/24	0	0	0	0	0
d_{12}.........	1/12	0	0	0	0	0

$$Z = a_0 \cos I - a_{10} \sin I \, dI + a_{20} \cos I \, dI^2 + b_{02} \cos I \sin^2 I \, dA^2$$
$$- a_{30} \sin I \, dI^3 + c_{12} \sin^3 I \, dI \, dA^2 + +d_{12} \sin I \cos^2 I \, dI \, dA^2 + O_4.$$

The results of this work have been applied to the following sequence of survey results, as shown in table 4.

As shown, the circular arc and the balanced tangential are very close to each other in departure results, owing to the proximity relation between their formulas. Note that although our results are given to six decimal places, the real world would suffice with two places, as the precision of data is certainly not better than two-place accuracy.

TABLE 4. - Example with survey data

Measured depth L, ft	Inclination angle I, degrees and minutes	Azimuth angle A, degrees
1,309.0	0 0	0 (N 0.0 E)
1,350.0	2 15	28 (N 28.0 E)
1,381.0	3 0	35 (N 35.0 E)
1,442.0	4 15	55 (N 55.0 E)
1,473.0	5 15	68 (N 68.0 E)
1,503.0	6 30	68 (N 68.0 E)

Bottom-hole departure results, in feet for programmable calculator and the six models:

Stations 1-6	X	Y	Z
Hand calculator circular arc....	8.89072	6.81338	1502.58396
Circular arc....................	8.89390	6.80903	1502.58398
Average angle...................	8.84176	6.97924	1502.58530
Balanced tangential.............	8.89347	6.80861	1502.57121
Radius of curvature.............	8.82279	6.[illegible]6123	1502.58061
Walstrom model 1................	8.85900	6.92236	1502.58061
Walstrom model 2................	8.78486	6.92520	1502.57121

The circular arc program has been implemented as a Fortran IV program with several support routines for the input-output (I/O) initialization and summarization of processed results. The package is usable for a time-share terminal. The rather elaborate I/O functions involve three files--one for the raw data, the second for the generated output data, and the third for a listing of processing errors. The error analysis is based on a modest verification applied to the raw input data, such as a check to see that the angular measurements are in an appropriate range of values or that the north-south and east-west indicators contain appropriate letters.

The circular arc method has been implemented also for use on the HP-67/97 and TI-58/59 by a graduate student at the Colorado School of Mines (John R. Henderson). Again, the same algorithm was used as in the previous program using the vector equations given previously. After initializing the coordinates and station number, the programs are quite flexible with regard to the input of data. The azimuth and quadrant information must be both entered. All the data must be entered before starting the program execution. Corrections can be made in any order without burdening the user, except that correct key entry must be used. Users of handheld computer systems will find documentation in the program writeups.

DRILL STABILIZER ANALYSIS

An important part of this research is the analysis of the placement of stabilizers in the drill string to control deviation. The research point of view is that there is an optimum placement of stabilizers with respect to a given drilling situation to achieve the desired bit-force and bit-axis directions. For a given drill string and rock formation, there is a complicated interrelationship between three basic parameters in any equilibrium drilling situation. These are the weight on the bit, the position of a single stabilizer with respect to the bit, and the position of the point of tangency with the hole above the stabilizer position. In employing even one stabilizer, a differential equation model is developed that has a complicated set of boundary value conditions.

It is theoretically easy, but an algebraically tedious task, to produce the set of approximate relationships which the drilling parameters satisfy. The appendix delves into this complicated subject to an elementary degree, and we will not attempt to offer mathematical proofs in this discussion. At this point we should note that our problem in raise-bore drill string mechanics is more amenable to successful analysis because of the extreme stiffness of the raise-bore drill string in contrast to other drilling, and we are dealing with small angles of drill-hole deviation and drill-string deflection.

The stabilizer problems that have been solved under this contract are shown in figure 2. They include one stabilizer for a near-bit stabilizer, and solutions for two, three, and four stabilizers. We will now briefly review the solution for the two-stabilizer problem, which we call the bottom-hole assembly, two-stabilizer, curved-hole problem. This problem will probably be one of the more useful programs for normal drilling conditions.

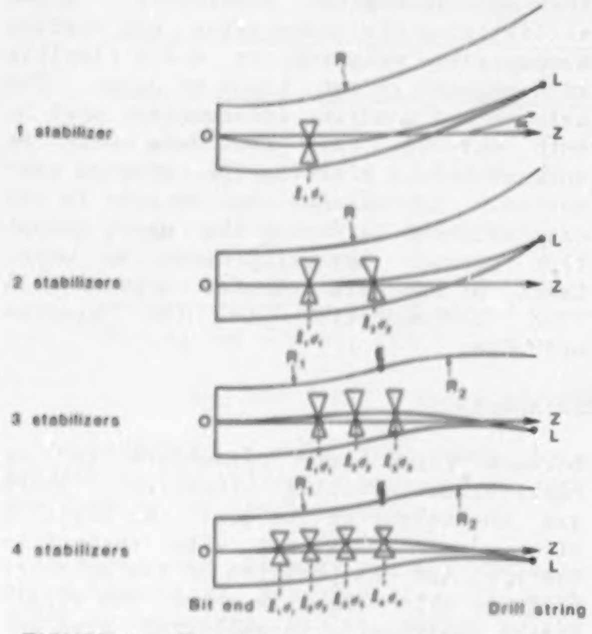

FIGURE 2. - Two-dimensional static stabilizer drilling programs.

As always in solving mathematical problems, there are certain assumptions, as follows:

1. The drill string behaves as an elastic structure.

2. The drill bit is centered in the borehole on the hole axis, and no moment exists at the bit-rock interface.

3. The components of the drill string are assigned arbitrary physical properties, which remain constant over some finite segment.

4. Displacements from the hole axis are small relative to the hole length.

5. The borehole walls are rigid and no deformation occurs.

6. Dynamic effects of the drill string and drilling fluid are neglected.

7. The drill string initially lies on the low side of the hole for some finite interval at some point above the pit.

The mathematical model is in the form of a fourth-order ordinary differential equation with boundary conditions, as above. In all our boundary value problems, the analyses were developed by looking at the solution of the approximate bending equation that satisfies the following two common boundary conditions at the bit; that is, $Z = 0$, $x(0) = 0$, and $x''(0) = 0$. The first conditions, $x(0) = 0$, simply means that the bit is centered in the bottom of the hole where $Z = 0$. The second condition, $x''(0) = 0$, means that the bit acts as a hinge against the bottom of the hole; or there is no bending moment exerted by the formation onto the drill string at the rock-bit interface.

As a solution to this particular boundary value problem for the previous bottom-hole assembly using two stabilizers in the curved-hole situation with stabilizer placement at distances of ℓ_1 and ℓ_2 above the bit, the solution form for the bending equation of the drill string can be represented by three expressions, one expression for each of the three segments of the lower section of the drill string. The partitioning of the interval for the integration of the ordinary differential equations introduces "conditions of continuity" at the stabilizer positions ℓ_1 and ℓ_2. These conditions will provide eight conditions of continuity given above, in conjunction with the three boundary conditions at the point of tangency, to produce 11 equations with 11 unknowns. These reduce, by the elimination of the linear relations, to a system of just three nonlinear equations, which are subsequently solved by the Newton-Raphson algorithm.

We caution that even this numerical technique may present problems in being able to make an initial estimate of the unknowns for which the computer will then make a precise solution of the problem. In a later section of this paper we will discuss the unique computer facility we have used for the solution to these problems.

The results of using the bottom-hole assembly, two-stabilizer, curved hole solution are shown for five drilling situations in table 5. Note that the hole is drilled 30° from vertical. In these examples we have used an 8-1/2-inch bit with 7-inch drill pipe having 2-1/4-inch ID. The industry unit weight for this drill pipe is 117.6 lb/ft (6). A suggested bit weight would be 40,000 lb for this size bit. As shown in table 1, we have provided five different sets of input positions for the stabilizers at ℓ_1 and ℓ_2 above the bit. The output lists the side force at the bit (H_0), the force at the first stabilizer (H_1), the second stabilizer (H_2), and the calculated point of tangency (ℓ) of the drill string above the bit. The important force vectors at the bit and the resultant bit angle of attack on the formation are also shown.

TABLE 5. - Five applications of the FORTRAN program

Position	Input, ft		Output, lb				Bit force angle, deg	Bit angle, deg
	ℓ_1	ℓ_2	H_0, lb	H_1, lb	H_2, lb	ℓ, ft		
20,000 POUNDS								
1..................	12	20	-454.5	-717.6	-842.9	43.9	-1.3019	0.0225
2..................	12	24	-700.2	-106.5	-1594.2	49.5	-2.0050	.0366
3..................	12	28	-769.2	-123.7	-1818.3	54.3	-2.2025	.0400
4..................	16	28	-672.0	-497.2	-1467.9	53.3	-1.9242	.0137
5..................	16	32	-724.7	-503.3	-1717.6	58.1	-2.0753	.0192
60,000 POUNDS								
1..................	12	20	-453.3	-688.8	-1029.9	43.0	-0.4329	0.0231
2..................	12	24	-695.5	-147.6	-1749.1	48.5	-.6641	.0374
3..................	12	28	-759.8	-209.1	-1965.4	53.3	-.7255	.0412
4..................	16	28	-689.3	-550.2	-1614.8	52.3	-.6582	.0154
5..................	16	32	-739.4	-596.6	-1858.5	57.1	-.7060	.0209

Input drilling parameters:

EI = flexural rigidity = 24,291,824.15 (lb/ft^2),

P = unit weight of drill collar = 117.6 (lb/ft),

α = inclination at bit form vertical = 30 (deg),

W = weight at bit in z-axis direction = 20,000 and 60,000 (lb),

DLS = dogleg severity = 1.0 (deg/100 ft),

B = radial hole clearance = 0.75 (inch),

D_1 = 1st stabilizer radial clearance = 0.167 (inch),

D_2 = 2nd stabilizer radial clearance = 0.375 (inch), and

ℓ = positions of stabilizers above bit and point of tangency (ft).

As a side benefit from this type of analysis, although it now seems intuitive, it became clear during this study that three conditions are necessary for straight-hole drilling. These are that the geometric axis of the hole, the geometric axis of the drill bit, and the resultant bit-force vector must all be coincident. Surprisingly, this simple fact has never been mentioned in previous literature.

The example problem with only 20,000 lb and 60,000 lb on the bit is shown in table 5. The reduced weight simulates the drillers logic that using less weight on the bit will drill a straighter hole. Running this problem over the range of 20,000- to 80,000-lb bit loading in 20,000-lb increments will show that changing weight is not all that significant. In fact, better bit loading and force vectors are obtained at higher bit loading than at lower bit loading for this particular example (see table 5). It is hoped that the lesson learned from these examples is that the sequence of stabilizer placement must be totally reevaluated and that an exact placement to the nearest foot be made. This is shown dramatically for stabilizer N_2 where side loads are approaching 2,000 lb, which is the industry maximum for avoiding drill-string fatigue. The methods of computer solution developed in this research offer the possibility of exact stabilizer placement and warnings of excessive side forces at the drill collars.

The computer program which we have used for the above two-stabilizer, curved-hole examples has been reduced to practice on an HP-41. Again, we have used a graduate student for this achievement (Ms. R. L. Callas). The handheld computer program offers the mining industry the ability to make drilling decisions right on the rig floor. Of course, these may be confirmed by the more comprehensive analog-digital program.

COMPUTER FACILITIES

We have referred to both analog and digital solutions in this paper. In reality, both solutions become necessary for the solution to the drill-string stabilizer problem. An EAI-2000 analog system is used by the Colorado School of Mines to make an approximate stabilizer placement solution, as well as an exact bending equation solution. The main DEC-1091 computer system of the school is used in a time-sharing mode with the EAI-2000 and executes Fortran computer programs to generate input coefficients and maximum modulus of particular solutions, along with the derivatives of the approximate bending equations. The extreme values are necessary to set amplitude scaling factors, which are necessary in graphing the differential equations by the analog computer. Another advantage gained by using the analog is in overriding the disadvantage of fixed-point arithmetic. The analog output is readily transferred to cathode-ray tube, or plotting routines, to continuously output the changing problem parameters.

It is the changing of parameters that is greatly facilitated by creating the hybrid configuration. Through a simple software interface, the operation of the EAI-2000 can be controlled by the digital DEC-1091. The analog-digital partnership shown in figure 3 can readily perform such ordinarily herculean tasks as changing input parameters and monitoring output results. For example, the hybrid configuration will automatically control boundary conditions for the exact solution of the bending equation. This control greatly enhances changing such parameters as weight on the bit and the subsequent automatic generation of approximate coefficient setting for the exact bending equation. With this aid in getting a handle on the problem, the approximate analog coefficient settings may be adjusted accordingly to satisfy the actual boundary conditions to within analog accuracy.

Another quite obvious advantage of the analog-digital hybrid solution is the

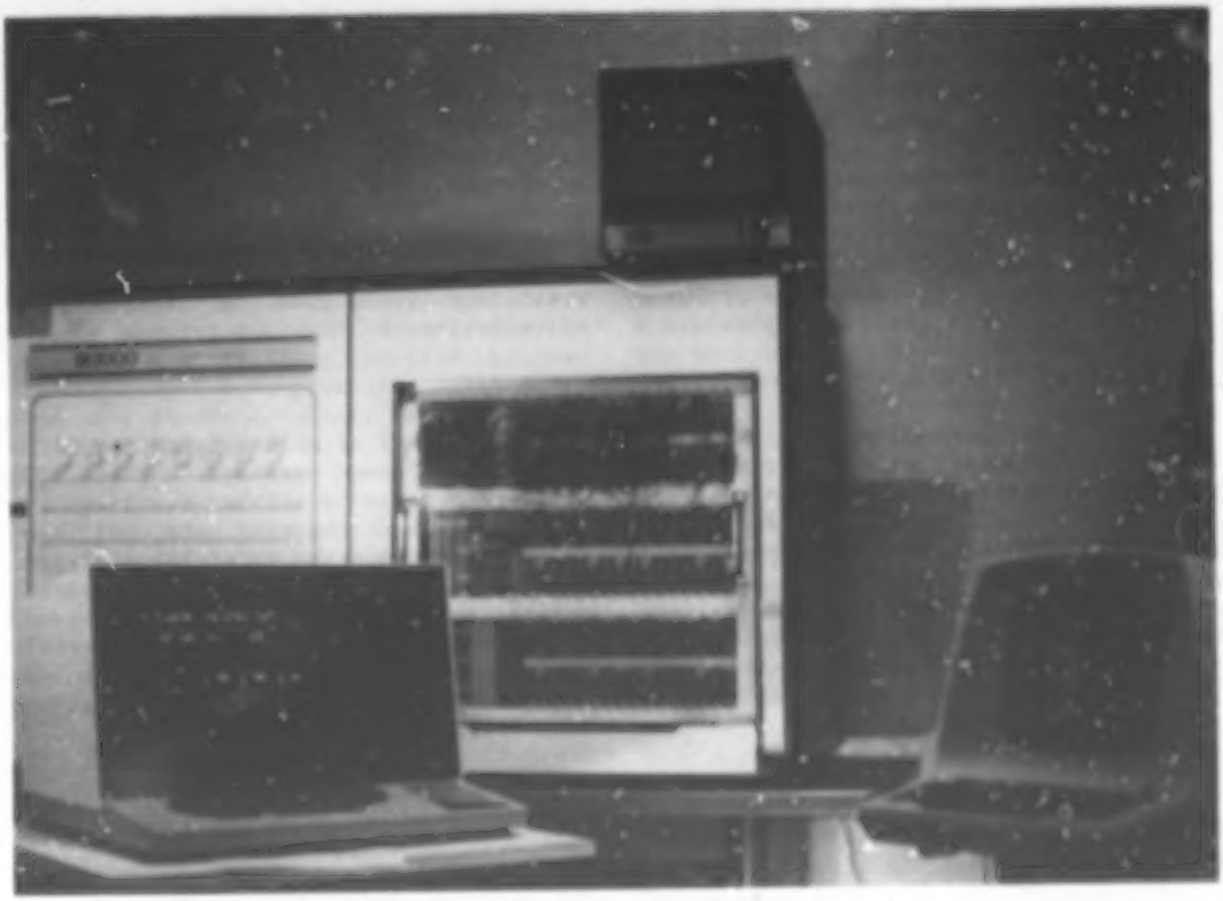

FIGURE 3. - Drilling analog-digital computer facility.

nearly instant visual inspection of the complete solution with given appropriate boundary conditions. Thus, the visual graphic solution gives instantaneous independent verification that both the Fortran digital program and the analog programs are giving correct solutions. A disadvantage of the analog system may be that it is not an exact solution. We have addressed this problem with the running of "benchmark-type" problems and our results to date suggest that the accuracy between the "approximate" analog solution and the "exact" digital solution may be on the order of 3%. Of course, it is pointed out that the drilling variables and the overall drilling requirements cannot be brought to much closer precision of calculation.

The analog-digital hybrid approach for solving the drill-string bending equations is apparently new. A variety of single valued solutions has been attempted over the past two decades. Lubinski (8), as well as others before, initially used a power series approach for solving the exact bending equations. Subsequently, iterative computer-type methods and other numerical integration techniques have been used. Recently, the trend of research appears to be the use of finite-element methods (10).

In contrast to the above methods, in particular the finite element, the analog-digital hybrid provides a complete solution at a very reasonable cost per run. Comparative cost figures are in the range of a few cents per run for the hybrid system versus up to a hundred dollars per run with the finite-element method, depending upon the complexity of the problem. The hybrid solution would likely compare in accuracy with the finite element for comparable types of drilling problems selected.

The analog-digital hybrid method of solving drill-string mechanics problems with stabilizers is new and represents a contribution to the mining industry by the Bureau of Mines. Indeed, these are the first complete solutions known for the stabilizer problem up to the complexity of four stabilizers, and the use of more than four stabilizers does not appear to offer any apparent drilling advantage. Of course, we welcome other investigators to continue the problem and a general solution up to the Nth stabilizer would be a mathematical contribution.

CONCLUSION

A conclusion reached by this study of borehole surveying practices and by optimum stabilizer placement analysis shows that there have been few documented cases of a geometrically straight hole drilled for any great depth. A perfectly straight hole can be defined as one in which the condition of geometrically straight is approached only as the limit under a well designed drilling program.

The drilling of a "straight" hole involves an optimum process. Thus, the hole size should be neither too large nor too small, and the size and weight of drill collars must be carefully analyzed. The placement of stabilizers in the drill string must be a carefully engineered decision. In summary, the straight-hole drilling program must be designed by competent engineers familiar with straight-hole drilling theory and practice.

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APPENDIX.--SIMPLIFIED DRILL STRING STABILIZER ANALYSIS

Economic considerations generally require that a drill hole be drilled at the lowest possible cost per foot. This is, of course, the natural drilling policy for the majority of industry. The least cost per foot depends upon, among other things, the average rate of drilling and the total feet drilled per bit. However, in relation to drilling a straight hole, these concepts may not necessarily be parallel. A drilling program planned with straight hole practices will optimize many economic factors entering into the drilling operation. It should also be remembered that the principal consideration is the end result. A drill hole requiring only several days or a few weeks to drill and complete may often involve an enormous expenditure.

Very little is actually known, in the way of scientific fact, about the precise mechanism of drilling. Available evidence indicates that the action of the bit is not independent of the overall characteristics of the drill string. For straight-hole drilling, the design and placement of the stabilizers have been shown to be one of the greatest single factors. Further evidence, although inconclusive, shows that the drilling action is influenced by the nature of the drilling fluid and the bit-mud pressure relations at the bottom of the hole.

The weight applied to the bit, the speed at which the bit is rotated, and the weight, size, and strength of the drill string are closely interrelated in rotary drilling. The drill collar at the lowest portion of the drill string above the bit should be considered as a bearing tool that holds the bit and orients it against the rock formation. As such, the weight, stiffness, mass distribution, and vibration characteristics of the entire drill string influence the action of the bit and the degree of stability with which it is held on the bottom. As an example of this stability, consider that when a 10-inch-diameter bit is loaded sufficiently to buckle the drill string, such that one side of the bit is lifted only 0.0135 inch off bottom, there will result a 1° tilt or bend in the drill collar (1, p. 62).

The relation between the weight on the bit and the rate of penetration has been well studied in both the laboratory and in the field. Of all the variables which affect the rate of penetration, the influence of the weight applied to the bit has been the most satisfactorily isolated and defined.

The weight on the bit is defined in drilling practice as the number of pounds of weight per inch of diameter and usually written as WPID. It is then a simplified measure of the weight per inch of bit area. General U.S. usage of the WPID index in field practice is about 5,000 lb of WPID. Use of higher WPID depends on the formation and is, as yet, inconclusive, but known to be desirable. The tendency in drilling practice since the 1950's is to increase the WPID due to improved material technology and increased understanding of the effect of WPID. Much of this WPID can be attributed to improved bit design, particularly bearings able to sustain higher WPID.

Less conclusive data on the effect of rotary speed (RPM) are available but, in general, increasing the WPID is more effective in increasing penetration rates than changes in RPM. The penetration rate has been found to be proportional to the first power of the rotary RPM, but proportional to between the 1.0 and 2.3 power of the WPID. Speeds of 30 to 75 RPM are preferred field practice to minimize joint failures and to improve bit tooth and bearing life. Drilling practice in the United States has tended toward slower rotary speeds with greater bit weights to obtain faster rates of penetration. Generally, as the weight on the bit is increased, the rotary speed is decreased.

However, the penalty for not using an optimum drill bit weight will be detrimental to the drilling contractor in any one of three ways. Low penetration rates and poor bit footage usually result with inadequate bit loading. Thus, the contractor is not obtaining economical rig use. Adding more weight to the bit than available will buckle the drill string and cause high tangential bit forces that cause extreme hole damage. The danger of pipe twistoff also becomes of increasing concern with excessive bit weight. When the load on the bit is further exceeded, the buckling forces are concentrated near the bit. The last penalty is then an increase in hole deviation.

Therefore, to stay within prescribed deviation limits, caution must be exercised against the use of indiscriminately high bit loads, especially in dipping formations. However, "holding up" on bit weight to prevent hole deviation generally results in reduced penetration rates and bit footages. These remarks are not intended to discourage the use of high WPID but rather to point out the necessity of intelligent planning for it.

The practice of straight-hole drilling then includes using optimal weight on the bit, heavier and more rigid drill collars, larger diameter stabilizers with less clearance between the stabilizer and the wall of the hole, and the precise placing of stabilizers at various positions along the drill string in order to control bending effects at the drill bit. In summary, the two most important drilling-related causes of hole deviation are the weight on the bit and bending of the drill string above the bit. Thus, it is seen that drilling a straight hole involves a unique optimization process.

Straight-hole drilling and the relationship between drill-string bending and hole deviation were placed on a mathematical basis by Lubinski and Woods in a series of papers beginning in the 1950's (7-8, 11-12). Their work is not the only literature reference to the term "straight-hole drilling practice." The petroleum industry as early as the late 1920's, through the American Petroleum Institute (API), established a committee for just such a purpose. The literature in this field is extensive, but credit must be given to Lubinski and Woods for the first attempt at solving this problem, and their work was only possible with the advent of electronic computers in the 1950's.

The physical analogy for this problem in drill-string mechanics can be considered from a rotating elastic column hinged at the ends and having both horizontal and vertical components. As such, the problem is basically not unlike the mechanical engineering solution for the deflections and thrust components for a length of line shafting subject to lateral loads. The design problem becomes in knowing where to place the bearings to minimize the deflection. Early authors considered that 6,000 ft of drill pipe had about the same elastic properties as 6 ft of coarse thread. A similar grasp of the elastic properties of the drill string is realized when it has been reported that treble stands of 4-1/2-inch, 16.60-lb drill pipe buckle into spaghetti right on the derrick floor (7, p. 211).

The following simplified analysis follows the method of Lubinski and Woods and is not intended as a comprehensive mathematical derivation of this very complex problem. For the initial analysis of the forces acting on the bit, we assume a slight initial hole inclination with the drill string lying on the lower side of the hole, except where it approaches the bit. It is also assumed that the drill string is unsupported at the point near the bit and the point of tangency where it comes in contact with the wall of the hole. The deviation angle of the hole is from the vertical, while another defined angle is in the direction of the force component tending to direct the hole away from the vertical. It is also intuited that at equilibrium, the condition is reached where the ratio of the two angles equals 1, and the hole deviation will remain constant. If we now isolate the drill bit into a free-body diagram and begin with the previous assumption that the drill string lies on the low side of the hole and makes contact with the wall only at the point of tangency, the forces with which the bit acts on the formation (frictional and rotational effects ignored) are applied at an angle with the vertical; the force at the bit may then be resolved into two components, a longitudinal force in the direction along the axis of the hole and a lateral force normal to the axis of the hole.

Three cases are proposed for the action of the forces in relation to the angles. It is seen from the force diagram that when the force acts on the low side: (1) the hole deviation will be decreasing; (2) the force may be zero and the hole deviation is in a stable condition; and lastly, (3) the force will act on the high side and the hole deviation will increase or we are building an angle. The amount of hole deviation being represented by the angle is subsequently shown to be dependent on three variables; the weight on the bit, the drill collar size, and the hole size.

The basis for deriving the bending relations of the drill string begins with the second-degree moment equation of the elastic curve for an unsupported member (7, p. 195). The derivative of the bending moment, of course, is the shearing force, A, as follows:

$$A = EI\,\frac{d^3y}{dx^3}. \qquad \text{(A-1)}$$

If the origin is now chosen from the bit end, and substitution made into the shear equation which, in turn, is written as the differential equation as follows (7, p. 198):

$$EI\,\frac{d^3y}{dx^3} + px\,\frac{dy}{dx} + F_2 = 0. \qquad \text{(A-2)}$$

Singular solutions to this equation were obtained by Lubinski in the 1950's using power series expansion and the method of iteration with an electronic computer. The preliminary work alone is extensive and we have only touched on the high points. The generally more important equations are given by Lubinski and Woods (8, p. 239, 11, p. 65, 12, p. 178).

By successive substitution and elimination of the various variables, the general equations may be solved by numerical methods. This was done by graphic form in the 1960's, and many sets of graphical and tabular data were developed for evaluating this important work of Lubinski and Woods.

The work of Callas (3) has expanded this beginning work into a series of solutions for various stabilizer configurations using the fourth-order ordinary differential equation with boundary conditions. The basic differential equation is of the form:

$$EI\,x'''(z) + Wx'(z) = H + zP\sin\alpha, \qquad \text{(A-3)}$$

where $z = 0$ and $0 < z < L$.

This is called the approximate bending equation by omitting the term $zP\cos\alpha x'(z)$ term in the exact bending equation. Justification for the omission of the term $zP\cos x'(z)$ from the exact bending equation is made under the assumption that the slope of the elastic line of the drill string is very small relative to the direction of the borehole at the bit. The term $H(z)$ on the right side is in reality an unknown integration constant and represents the unknown forcing function at the points of contact of the drill string with the hole at the bit and the stabilizer position below the point of tangency. The approximate bending equation has been solved analytically, in closed form, for a number of different sets of drilling conditions, representing various stabilizer configurations. In all cases, the general solution form of the approximate equation is the following:

$$x(z) = K_0 + K_1\cos\left((W/EI)^{1/2}z\right) + K_2\sin\left((W/EI)^{1/2}z\right) + (H/W)z + (p\sin\alpha/(2W))z^2, \qquad \text{(A-4)}$$

where K_0, K_1, K_2, and H are arbitrary constants. The values of these integration constants depend upon the boundary conditions which are introduced in a particular problem.

From drill string mechanics, the side forcing function H takes the form:

$$H = \begin{cases} H_0,\ z = 0 \text{ or } 0 < z < l_1, \\ H_0 + H_1,\ l_1 \leqq z < l_2, \text{ and} \\ H_0 + H_1 + H_2,\ l_2 \leqq z < l_3, \\ \text{etc.} \end{cases} \qquad \text{(A-5)}$$

The term H_i represents the side forces on the respective stabilizers, however many there are in the drill string assembly, $i = 1, 2, \ldots, n$. The stabilizers are considered as "point" sources of support throughout these analyses.

In all of our boundary value problems, the analyses were developed by looking at the solution of the approximate bending equation which satisfies the following two common boundary conditions at the bit, $z = 0$ and $x(0) = 0$ and $x''(0) = 0$.

The first condition, $x(0) = 0$, simply means that the bit is centered in the bottom of the hole, $z = 0$. The second condition, $x''(0) = 0$, means that the bit acts as a hinge against the bottom of the hole; that is, there is no bending moment exerted by the formation to the drill string at the rock-bit interface. The solution of the approximate bending equation, using just these two conditions, is as follows:

$$x(z) = (EI\ p\sin\alpha/W^2)\ [\cos((W/EI)^{1/2}z) - 1] + K_3\sin((W/EI)^{1/2}z) + (H_0/W)z + (p\sin\alpha/(2W))z^2, \qquad \text{(A-6)}$$

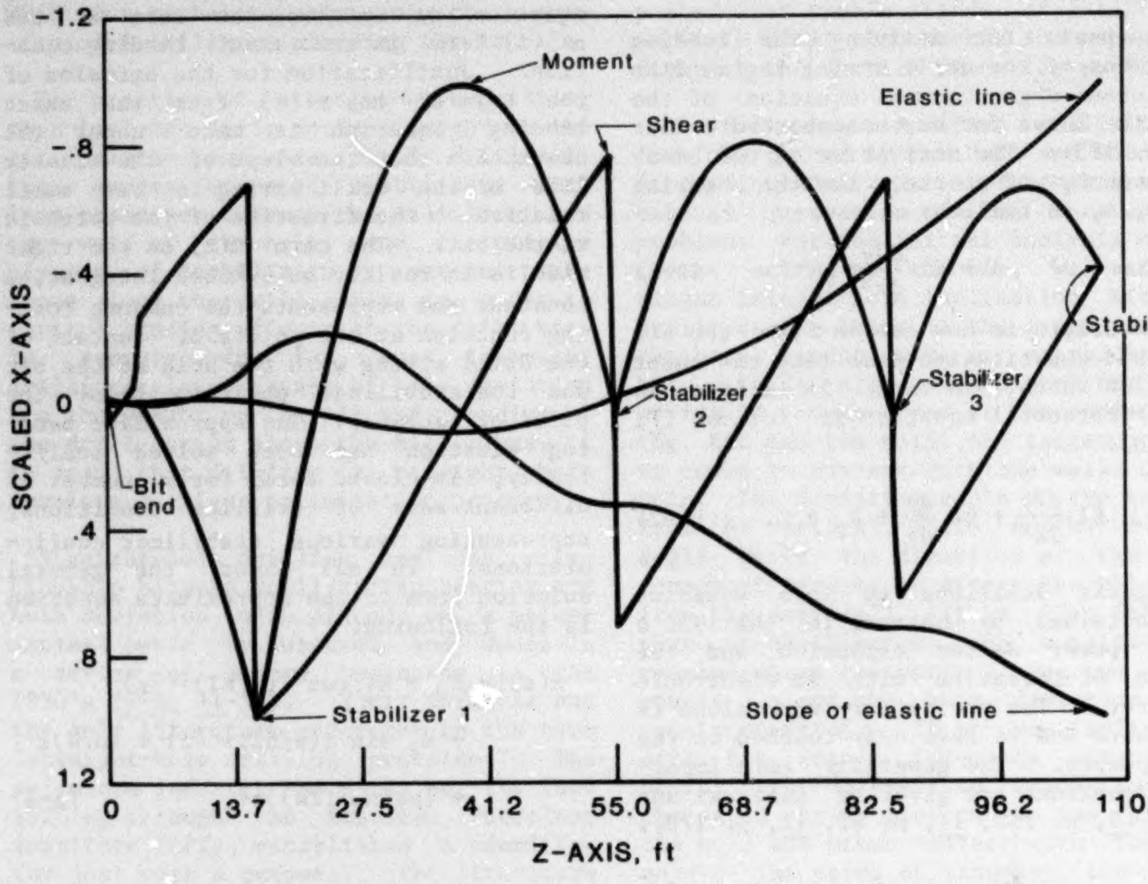

FIGURE A-1. - Analog solution for typical four-stabilizer drill-string problem.

where K_3 and H_0 are still unknown integration constants. Once the numeric values to these constants are calculated, they are used to compute the values of the coefficients for each of the expressions described for the given segments of the drill string. The side forces, H, at the bit and the stabilizers are then easily obtained. The complete solution for a typical two-stabilizer problem is illustrated in figure A-1 for stabilizers placed at 12 and 20 ft above the bit.

These results can be readily translated into an economic understanding of the drilling consequences. An increase of weight on the bit increases the bending of the unsupported portion of the drill string above the bit that moves the point of tangency closer to the bit and increases the force causing hole deviation. Therefore, uncontrolled weight on the bit can result in increased hole deviation. The size of drill stabilizer and size of hole are two factors interrelated through their mutual effect on hole clearance, which is the difference between the hole diameter and the drill stabilizer diameter. Correctly sized and placed stabilizers, being heavier and less likely to bend, will move the point of tangency further away from the bit. A stabilizer, or series of stabilizers, which serve as bearing points is used as a means of controlling the location of this contact point. When the effective point is moved up the hole, the hole straightening force imposed by the weight on the bit is increased. A series of properly positioned stabilizers above the drill bit will allow a larger WPID to be used which will give better rate of penetration, more economic rig use, and a straighter hole.

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